I. C. 6500

May, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXV

BY

Frederick W. Lee

I. C. 6500
May, 1931

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 25

Compiled by Frederick W. Lee[2]

List of contributing editors of Geophysical Abstracts:

Ayvazoglou, W., U. S. Bureau of Mines, Department of Commerce, Washington, D. C.
Barton, Dr. D. C., Petroleum Building, Houston, Tex.
Belluigi, Dr. Arnaldo, Corso Vittorio Emanuele 178, Parma, Italy.
Bogoiavlensky, Prof. L., Central Chamber of Weights and Measures, Leningrad, U.S.S.R.
Eckhardt, Dr. E. A., 327 Craft Ave., Pittsburgh, Pa.
Eve, Dr. A. S., McGill University, Montreal, Canada.
Gish, Dr. O. H., Carnegie Institution, Broad Branch Road, Washington, D. C.
Gorsky, Eng. V., Allatini Mines, Ltd., Skoplie B.p. 134, Yugoslavia.
Hartley, Kenneth, 2404 San Jacinto St., Houston, Tex.
Hutchinson, Prof. W. Spencer, Mass. Institute of Technology, Cambridge, Mass.
Jenny, Dr. W. P., Magnolia Petroleum Co., Dallas, Tex.
Karcher, Dr. J. C., Dallas, Tex.
Keys, Dr. D. A., McGill University, Montreal, Canada.
Knappen, Dr. R. S., Gypsy Oil Co., Tulsa, Okla.
Korzujin, Prof. J., National University of Mexico, Mexico, D. F.
Lane, Prof. Alfred C., Tufts College, Boston, Mass.
Lee, Dr. F. W., U. S. Bureau of Mines, Department of Commerce, Washington, D. C.
Leonardon, E. C., 25 Broadway, New York City.
Numerov, Prof. Dr. B. V., Fontanka 34, Leningrad, U.S.S.R.
Petrowsky, A., Wasilly Ostrov, 21 Linia No. 3-A, Leningrad, U.S.S.R.
Roman, Dr. I., 90 Valley Way, West Orange, N. J.
Ruark, Dr. A. E., University of Pittsburgh, Pittsburgh, Pa.
Scholl, Louis A., Box 1805, Houston, Tex.
Shaw, Dr. H., The Science Museum, South Kensington, London, S.W. 7.
Sundberg, Dr. Karl, Swedish American Prospecting Corp., 26 Beaver St., N. Y. C.
Truemann, O. H., Humble Oil Co., Houston, Tex.
Van Orstrand, Dr. C. E., Interior Building, Washington, D. C.
von Weelden, Dr. A., Shell Petroleum Corp., Dallas, Tex.
Weaver, Paul, Drawer C, Houston, Tex.
Wright, Dr. F. E., Carnegie Institution, Washington, D. C.
Zuschlag, Dr. Theodor, Swedish American Prospecting Corp., 26 Beaver St., N. Y. C.

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2 - Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(162) CONTRIBUTION TO THE QUESTION OF THE DISPLACEMENT OF THE ZERO POINT OF THE TORSION BALANCE IN A GRAVITY VARIOMETER (IN RUSSIAN)

By D. G. Uspenski

Transactions of the Institute of Applied Geophysics, Leningrad, U.S.S.R., vol. 6, 1930, pp. 3-10.

In this article the author gives the results of the tests carried out in the Gravimetrical Laboratory of the Institute of Applied Geophysics concerning the temperature effects upon the displacements of the beams of a gravity variometer.

The tests were performed under conditions similar to those described in the previous work of the author published under the same title (see Geophys. Abs. No. 9, p. 5). By working with instruments devised by the Institute of Practical Geophysics, the state of equilibrium of the beam can be established by measuring the damping sinusoids registered by the beam.

The results of the tests are considered to be successful.--W. Ayvazoglou.

(163) OBSERVATIONS WITH GRAVITY VARIOMETERS AT THE SLAVIANSK SALT DEPOSITS IN 1928-29 (IN RUSSIAN)

By D. G. Uspenski

Transactions of the Institute of Applied Geophysics, Leningrad, U.S.S.R., vol. 6, 1930, pp. 11-27.

These observations were carried out in the winter of 1928-29.

Contents of the article:

1. Instruments and accessories.

2. Observations with gravity variometers.

3. Efficiency of the instruments and analysis of failures.

4. Shift of the zero point and accuracy of instruments.

Two gravity variometers constructed by the Seismological Institute, Academy of Sciences, and by the Institute of Applied Geophysics (Type 1928, Nos. 239 and 240) were used.

The author gives a series of tables in which the work of Russian instruments is compared with Schweydar-Bamberg's model and Hecker's large model (the data for the latter two being taken from the observations made in Krivoy Rog in 1927).

In conclusion the writer says that, based on the comparative data given, the Russian instruments of the 1928 type must be considered more suitable than Bamberg's and Hecker's instruments, as they have shown higher efficiency in spite of more difficult conditions of work.--W. Ayvazoglou.

(164) THE QUANTITY $\frac{\partial g}{\partial z}$ AND ITS ROLE IN GEOPHYSICS

By M. A. Sadovsky

Transactions of the Institute of Applied Geophysics, Leningrad, U.S.S.R., vol. 6, 1930, pp. 28-47.

This paper deals with the possibility of measuring the quantity $\frac{\partial g}{\partial z}$ the anomalous gradient of gravity tension along the vertical, and of applying the said gradient to the solution of problems concerning mass distribution in the upper layers of the earth crust. It has not been so far possible to determine this quantity with a sufficient accuracy, although the knowledge of it would be of a very great help for the solution of many practical as well as theoretical problems.

The following principal cases in which the quantity $\frac{\partial g}{\partial z}$ could be applied are mentioned:

1. Reduction of pendulum observation.

2. The solution of the problem on the structure of the upper layers.

3. The definition of the mean density of the earth and of the shape of the geoid.

From the mathematical calculations given in this article the author concludes that the technical realization of such an instrument is impossible. Moreover, such an apparatus would probably possess numerous defects which would considerably reduce the range of its application.--W. Ayvazoglou.

(165) A BRIEF ACCOUNT OF THE RESULTS OF GRAVIMETRICAL PROSPECTING AT THE PRIMARY PLATINUM DEPOSITS IN THE DISTRICT OF NIJNI-TAGIL

By P. M. Nikiforov and S. K. Ghirin

Transactions of the Institute of Applied Geophysics, Leningrad, U.S.S.R., vol. 6, 1930, pp. 48-58.

The purpose of the investigation described in this article may be summed up as follows:

1. To establish the outlines of the platinum-bearing dunite massif among the rocks surrounding it.

2. To determine the thickness and the position of the massif.

3. To isolate the parts of the dunite massif containing chromite as the latter forms the main primary platinum deposits.

Observations were made with instruments constructed in the workshops of the Seismological Institute of the Academy of Sciences. Two hundred and seventeen observations were taken.

This paper deals with the characteristics of physical values obtained and contains, besides, some conclusions drawn from the study of the whole material.

The anomalous gradient values of the force of gravity are shown on the annexed diagrams.

The authors conclude that their attempt to study the geological problem by gravimetrical method has yieldod good results and all the questions, in spite sometimes of schematical solutions given, have been answered. The work will be continued in order to clear up the points of the problem not sufficiently disclosed.--W. Ayvazoglou.

2. MAGNETIC METHODS

(166) REMANENTER MAGNETISMUS UND GESTEINSFLUIDITÄT

(REMANENT MAGNETISM AND THE MAGNETIC FLUX OF ROCKS)

By J. Koenigsberger

Zeitschrift fuer Praktische Geologie, Halle,
vol. 39, No. 2, 1931, pp. 26-27.

In this article the author gives a comparison of the results of experiments carried out by different methods on the determination of the susceptibility and the remanent magnetism of rocks. The methods discussed are those used by K. Puzicha, G. Grenet, and J. Koenigsberger (see Geophys. Abs. Nos. 22, p. 4; 20, p. 5; and 18, p. 4, respectively).

Taking into consideration the fact that samples of rocks taken from the same quarry often display differences in susceptibility, the remanence and the ratio of the remanent magnetism to the induced magnetism amounting sometimes to 100 per cent, the author says that the purpose of his investigation does not concern the accuracy of the methods applied but merely the question of the possibility of quick measurements without making systematic errors.--W. Ayvazoglou.

(167) DIE MAGNETISCHEN GRANITE VON SCHIERKE IM HARZ

(THE MAGNETIC GRANITES OF SCHIERKE IN THE HARZ)

By Kurt Puzicha

Centralblatt fuer Mineralogie, Geologie und Paläontologie, Stuttgart,
Abt. B., No. 1, 1931, pp. 1-6.

The displaying by rock layers of self-magnetism produced by lightning strokes, in addition to their magnetization by the influence of the earth's

field, may sometimes be the cause of incorrect conclusions on magnetic anomalies in certain regions, as determined by magnetic surveys.

In order to find out to what extent the magnetic anomaly observed in the region of the granites of Schierke may be influenced by this self-magnetization of the granites the author carried out laboratory investigations of eight samples taken from the border zone of the Brocken-massif and proved that the susceptibilities of these granites were sufficiently high to cause the magnetic anomalies established by Reich.

The results of investigations are described in this article and illustrated by two diagrams. Numerical data are given in a table.--W. Ayvazoglou.

(168) ATMOSPHERIC PRESSURE AND THE STATE OF THE EARTH'S MAGNETISM

By J. M. Stagg

Nature, London, vol. 127, No. 3202, 1931, p. 402.

The author discussed the obtained results pointing to a hitherto unsuspected relation between the type of the diurnal variation of pressure and the general state of magnetic conditions, as regards disturbance and quiet over the earth.

Mean diurnal inequalities of pressure corrected for nonperiodic change were computed for the two sets of days, magnetically quiet and magnetically disturbed, on the basis of the five days of each type per month over the seven years, 1922-1928. By grouping the inequalities according to the season of the year, diurnal variations of pressure were constructed. A further rearrangement of the data gave inequalities illustrating the pressure variations on the two types of days in years of low and high solar activity.

The results are shown in the six pairs of curves, one of which represents the diurnal variation of pressure on magnetically quiet days and the other on magnetically disturbed days. The difference in form of the curves shows the change in the pressure variation from one type of magnetic days to the other.

From these it is clear that in all but the last pair the predominant features of the change are the reduced development of the forenoon maximum and enhancement of the evening maximum on disturbed as compared with quiet days.--W. Ayvazoglou.

3. SEISMIC METHODS

(169) SEISMIC METHODS OF PROSPECTING AS APPLIED IN MINING (IN RUSSIAN)

By Dr. Heise

Gorny Journal (Mining Journal), Moscow,
vol. 106, No. 6-7, pp. 163-166.

After a brief description of the principles of seismic methods of prospecting Heise mentions the great success obtained by this method in prospecting for oil in America, especially in Texas and Louisiana. A map showing, for comparison, the salt domes discovered in these States by geophysical methods, and those known before the introduction of these methods is given.

The possibility of determination by the seismic method of anticlinos, faults and similar irregular dislocations of rock masses makes the application of this method in mining important. In a figure the author gives the contour lines of the region of coal deposit in the Government mines of Hendrik, near Heerlen in Holland, as established by the seismic survey. The survey shows that the relief of the surface of the coal deposit is not as level as could be expected on the basis of the results obtained by drilling a few holes.--W. Ayvazoglou.

(170) BEITRAG ZUR BERECHNUNG VON KONSTANTEN DER GALITZIN'SCHEN APERIODISCHEN SEISMOGRAPHEN

(CONTRIBUTION TO THE CALCULATION OF CONSTANTS OF GALITZIN'S APERIODIC SEISMOGRAPHS)

By E. Büss

Académie des sciences de l'U.R.S.S., Leningrad, Publication de l'institut séismologique, No. 8, 1930, pp. 1-11

Formulas are derived for working out special tables which may serve for finding the accurate values of the constants during the observations without considerable calculations, so that the necessary adjustments may be made without delay. The tables are added to the article.--W. Ayvazoglou.

(171) SEISMOLOGY AND ENGINEERING

By N. H. Heck

The Military Engineer, Washington, D. C.,
vol. 23, No. 128, 1931, pp. 131-134.

The effect of artificial earthquakes produced by means of large explosions and the outline of the fundamental facts regarding earthquakes, as well as the progress that has been made in understanding them are briefly discussed.

The principles of the construction of seismographs are described and typical seismograms and photographs of various seismographs are given.

The greatest importance of earthquakes from the engineering viewpoint with regard to the visible effects on the earth's surface may include: Fracture along a fault plane with horizontal or vertical slipping along a vertical plane; slipping along an inclined plane; wholesale raising of an area; wholesale lowering over a large area; and movement of large blocks, either tilting or raising or lowering.

Thus the effects on structures are very complex.

The study of these effects artificially produced by vibrating platforms (Stanford University) for obtaining valuable information is mentioned.

The author concludes the article by enumerating the organizations working on the general problems connected with seismological research.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(172) CALCULATION OF THE POSITION OF THE REFLECTING SURFACE ACCORDING TO OBSERVATIONS MADE BY MEANS OF THE RETURN METHOD (IN RUSSIAN)

By A. Petrowsky

Transactions of the Institute of Practical Geophysics, Leningrad, U.S.S.R., vol. 5, 1930, pp. 3-27.

This paper is a continuation of Petrowsky's article, "Theory of the return method" (see Geophys. Abs. No. 1), and is based on the formulas deduced in it. The purpose of this article is to find practical methods by means of which it would be possible to answer the following questions:

1. How is the reflecting surface situated with regard to the vibrator sending the waves?

2. At what distance from the center of the vibrator is this surface situated?

To answer the first question it is necessary to make a series of observations of the generation under different positions of the vibrator, that is, under different angles of inclination of the axis of the vibrator to the reflecting surface. The study of these waves makes it possible to determine the direction in which the vibrator is perpendicular, as well as the directions in which the vibrator is parallel to the reflecting surface.

The answer to the second question may be obtained from the same generation curves but also in the following way: The vibrator can be given a certain definite position--for example, parallel to the reflecting surface--and then generation curves of different lengths of waves may be drawn.

Mathematical details, the knowledge of which is necessary for a correct calculation of the results, are given.

Diagrams and tables complete the article.--W. Ayvazoglou.

(173) BASES FOR THE CALCULATION OF THE EQUIPMENT REQUIRED FOR ELECTROMETRICAL PROSPECTING (IN RUSSIAN)

By A. Petrowsky

Transactions of the Institute of Practical Geophysics, U.S.S.R., Leningrad, vol. 5, 1930, pp. 28-41.

The author makes an attempt to utilize the results of theoretical works on electrical prospecting for the determination of conditions under which the equipment necessary for this prospecting could be used with the greatest advantage, avoiding all unproductive expenses.

All the calculations are made with regard to a deposit having a spherical shape. The author applies formulas deduced by him in his previous articles: "Natural Electric Field Produced by Ore," and "Determination of the Location, Depth and Thickness of a Spherical Ore Body by Observing the Earth Current Produced" (see Geophys. Abs. No. 1, pp. 13 and 14).

The following questions are discussed:

1. The choice of the dimensions of the field explored and the thickness at the points of observation. The longitudinal dimension of the field must be not less than four times the depth of the center of the spherical ore body and the thickness at the points of observation not less than two-tenths of this depth. Transversal dimensions in the case of a spherical form of the ore body must be equal to the longitudinal ones, thus the application of a square field with 21 points in both longitudinal and transversal directions is most reasonable.

2. Two kinds of prospecting: The experimental--intended only to discover the presence of the ore body; and the detailed--to find the position of the ore body and the other elements characterizing it.

Mathematical calculations for the determination of the magnitude of the deposit by observations made on the natural electric field and calculations on the current strength, intensity and power of the equipment necessary for the observations upon the artificial electric field are given.

The author draws the conclusion that the difficulties of electrometrical prospecting increase quickly with the depth of the ore body, therefore there is no reason to rely upon this method in case of very great depth--

for example, depths reaching a few kilometers. The calculations are applied to an example given at the end of the article.—W. Ayvazoglou.

(174) CALCULATION OF THE GALVANOMETRIC SET FOR THE MEASUREMENT OF THE GRADIENT (IN RUSSIAN)

By A. Petrowsky

Transactions of the Institute of Practical Geophysics, Leningrad, U.S.S.R., vol. 5, 1930, pp. 42-44.

As basis for the calculations given in this article the theoretical formulas derived by the author in his articles: "Theory of the measurements of earth currents" and "Bases for calculating the observations of earth currents" (see Geophys. Abs. Nos. 1 and 2) are used. Calculation of the sensitiveness of a galvanometer under which it becomes unsuitable for the work is the purpose of this article. Petrowsky concludes that: "For the work on a damp ground ($\rho = 10^4\ \Omega\ cm^2/cm$) it is necessary to use a galvanometer with the sensitiveness equal to $0.4/\mu$ A, while on a dry ground ($\rho = 10^5 \Omega\, cm^2/cm$) this sensitiveness must be equal to $0.04/\mu$ A."--W. Ayvazoglou.

(175) ONDOMETRICAL WORK CARRIED OUT IN THE INSTITUTE OF PRACTICAL GEOPHYSICS (IN RUSSIAN)

By A. Petrowsky, R. Skariatin, A. Seleznev, B. Dostovalov and L. Kleiman

Transactions of the Institute of Practical Geophysics, Leningrad, U.S.S.R., vol. 5, 1930, pp. 45-69.

The beginning of the ondometrical work (i.e. the application of electromagnetic waves to ore prospecting) took place in 1923. During that year and the following the first portable frame transmitter of small power, intended for ore prospecting, which worked according to Meissner's scheme, was constructed in collaboration with the Leningrad Electrotechnical Experimental Laboratory (figures of the apparatus are given).

The experiments with this transmitter, which had been carried out in the Ilezk salt mines during the summer of 1925, proved the possibility of radio connection through the salt at small distances (about 100 to 150 meters) by means of a pair of such sets.

The experience obtained resulted in the construction of a new transmitter which was tried out in the Ilezk salt mines during the summer of 1927.

In 1928 there was constructed a new receiver (shown in two figures). An expedition to the Tchiraghi-Dzor mines, Caucasus, was equipped with these instruments. The aim of the expedition was to find out the possibility of transmitting the signals through the quartz rocks and the application of the shadow-method to the discovery of the pyrite bodies. The work was carried out as follows: The sender was placed within a drift and the receiver was moved from point to point on the day-surface.

The intensity of the signals received was read at every point by means of a galvanometer. Experiments were made also, with the receiver placed inside of the drift and the transmitter on the day-surface.

The following results were obtained from these experiments:

1. The possibility of the radio transmission of electromagnetic waves, 50 to 250 meters in length, through quartz rocks must be considered to be proved definitely.

2. Radio transmission took place in both direct as well as inverse directions.

3. An ore body (pyrite) placed behind the sender had no marked influence upon the radio transmission.

4. An ore body (pyrite) placed in the path of the electromagnetic waves produced an electromagnetic shadow.

During the tests the authors established the influence of the length of the wave upon the intensity of the signals. In several cases a clear minimum was observed; the reason for this minimum could not be established but it was curious that the length of the wave to which this minimum corresponded was almost equal to four times the average height of the trees surrounding the drift in which the transmitter was operated.

Thirty figures illustrate the article.--W. Ayvazoglou.

(176) A NEW DEVELOPMENT IN ELECTRICAL PROSPECTING

By Hans Lundberg and Theodor Zuschlag

The American Institute of Mining and Metallurgical Engineers, New York, Technical Publication 415, 1931, 18 pp.

A new measuring procedure permitting potential drop ratio determinations is described. After a brief description of the classification of electrical methods into qualitative and quantitative ones, and the further subdivision of the latter into potential drop and potential ratio techniques, the authors point out the advantages which would accrue from the elimination of certain deficiencies in potential ratio studies. These deficiencies are due to the existence of unknown and variable contact resistances at the earth connections. These contact resistances may be eliminated by a mathematical procedure which was made the basis of the design of the potential drop ratio compensator called the "Racom," developed by Zuschlag.

Generally speaking, the Racom is a sensitive bridge arrangement for the accurate and speedy execution of potential drop ratio determinations, and is thus adapted to geoelectrical exploration. The principles of its operation and use are described with reference to a series of figures. The results of four actual investigations, including ore hunting, determining the depth of water and of bed rock and depicting the underlying geological columns at various stations of interest to the oil geologist are discussed.--W. Ayvazoglou.

(177) HUNDERT JAHRE ELEKTRISCHE BODENFORSCHUNG

(ONE HUNDRED YEARS OF ELECTRICAL PROSPECTING FOR ORE)

By B. Duschnitz

Kali, Berlin, vol. 25, Nos. 5 and 6, 1931, pp. 71-76 and 88-92.

Duschnitz gives an interesting sketch of the development of the electrical methods of prospecting since 1830. Beginning with the first experiment carried out by Robert W. Fox (Philosophical Transactions, 1830, pp. 399-414), and after having mentioned the names of a series of other investigators, the author gives a more detailed description of prospecting for ore carried out by electrical waves according to Trüstedt (1901).

The methods originated by Löwy and Leimbach, called by them: (1) The reflection method; (2) the absorption method; (3) the capacity method; (4) the interference method; and (5) the method of quarter waves (Viertelwellenmethode) are discussed, their application at different places is described and conclusions are drawn.

All the electrical methods of prospecting are divided by the author into three main groups:

1. Prospecting by means of Hertzian waves.

2. Prospecting by means of natural earth-currents.

3. Prospecting by means of artificial currents produced in the ground.

Although all the three groups supplement one another, the third group is considered the most important one, as these methods make possible the obtaining of a graphic picture of the area under investigation.

The application of the electrical methods of prospecting by Schlumberger, Ambronn, Heine, Krahmann and many others is mentioned.

A long list of literature is added.--W. Ayvazoglou.

(178) A THEORETICAL STUDY OF APPARENT RESISTIVITY

By J. N. Hummel

American Institute of Mining and Metallurgical Engineers, New York, Technical Publication 31, 1931, pp.

A number of surface potential methods and the relation of potential distribution to subsurface resistivity are discussed. The principal field of application of these methods is the investigation of strata with horizontal (or nearly horizontal) bedding planes. Absolute potential differences are measured along a line at the surface. Of all possible arrangements of electrodes, five are listed which are most suitable from the standpoint of

field procedure and clearness of interpretation. From the potential differences, the subsoil resistivity, termed "apparent resistivity," may be computed. It changes with the distance between the secondary electrodes and is also a function of the resistivities of the surface formations as well.

The cases of a spherical mass, a 2-layer problem, and a 3-layer problem, are analyzed. In the layer problems, the author makes use of the theory of images.

From the potential, the expression for the apparent resistivity is derived as a function of the distance between the secondary electrodes. Several curves are given for different ratios of resistivity. In the 3-layer problem, an approximative curve is computed which approaches the true curve for large distances, and the true curve can be found by graphical interpolation. This procedure greatly simplifies the mathematical analysis.--Abstract from Mining and Metallurgy, vol. 12, No. 292, 1931.

(179) ELECTROMAGNETIC ABSORPTION BY ROCKS WITH SOME EXPERIMENTAL OBSERVATIONS TAKEN AT THE MAMMOTH CAVE OF KENTUCKY

By J. Wallace Joyce

U. S. Department of Commerce, Bureau of Mines, Technical Paper 497, 1931, 28 pp.

This paper endeavors to shed some light on the question of the penetration of electromagnetic fields or waves into the ground. It is intended to settle definitely the fact that such waves penetrate the earth's surface and that, in so doing, absorption occurs. The question is vital to geophysics, since the successful operation of all induction methods depends upon this factor.

Contents of the article:

1. Fundamental principles of induction methods.

2. Factors influencing reaction of conducting bodies.

3. Absorption formulas (Maxwell's, Zenneck's, King's and Sommerfeld's absorption formulas).

4. Comparison of absorption formulas.

5. Theory of method of experimental measurement of absorption.

6. Formulas for calculation of coefficient of mutual inductance between two circular loops with planes parallel and not in the same plane.

7. Experimental work (early work, site of work, preliminary tests, qualitative work, quantitative work, procedure, experimental results).

8. Discussion of results.

In conclusion the author says that the fact that electromagnetic waves actually penetrate rock has been definitely established. A start has been now made toward the quantitative verification of absorption formulae, and with the material contained in this paper as a foundation, additional experimental facts will be obtained in the near future which will no doubt provide a great additional amount of valuable information.

The results of this investigation show that a frequency of 500 cycles per second is well suited for electromagnetic prospecting. Absorption, although present, does not materially limit the applicability of this method.
--W. Ayvazoglou.

(180) EARTH-RESISTIVITY MEASUREMENTS IN THE COPPER COUNTRY, MICHIGAN

By W. J. Rooney

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, Md., vol. 32, No. 3/4, 1927, pp. 97-126.

During the summer of 1927 the Department of Terrestrial Magnetism of the Carnegie Institution and the Michigan College of Mines and Technology jointly conducted a series of earth-resistivity measurements in the copper country of northern Michigan, where the geological structure is well known, to determine the value of measurements of resistivity and its variations as indications of the geological structure below the surface. The work done may be roughly divided in two parts: First, measurement and study of the variations in resistivity with depth, or with volume, to establish such general relations as exist between the two; and, second, the determination of the resistivity of specific rock-flows, in order that more fundamental data be made available as an aid to the interpretation of the results of the former type of measurement.

The results of the first part of the investigation bear out the evidence from previous work that resistivity-determinations may be used to advantage in determining the location of certain underground discontinuities. The data obtained in the second part on the resistivity of specific rocks are also in line with previous results. The resistivity of porous, sedimentary rocks was usually under 20,000 ohms per centimeter cube, while traps and associated rocks of volcanic origin were found to have resistivities from five to twenty times as great.--Author's abstract.

5. RADIOACTIVE METHODS

(181) A MORE ACCURATE AND MORE EXTENDED COSMIC-RAY IONIZATION-DEPTH CURVE, AND THE PRESENT EVIDENCE FOR ATOM-BUILDING

By Robert A. Millikan and G. Harvey Cameron

The Physical Review, Minneapolis, Minn., vol. 37, No. 3, 1931, pp. 235-252.

The cosmic-ray ionization-depth curve has been extended at both its upper and lower ends and made more accurate throughout. The absorption

coefficients obtained directly from the slope of the curve run from $\mu = 0.35$ per meter of water at the top (Pike's Peak) to $\mu = 0.028$ at the bottom (80 meters or 262 feet of water below the top of the atmosphere), thus bringing to light both softer and harder components than the authors had before found. Strong quantitative evidence is presented, on the basis of the Klein-Nishina formula, that the strongest and most absorbable cosmic-ray band arises from the act of formation of helium out of hydrogen. Striking quantitative evidence is found that the three more penetrating bands are due to the formation out of hydrogen of the only other abundant elements oxygen (C, N, O) silicon (Mg, Al, Si, S) and iron (iron group). Two independent proofs are given that the cosmic-rays enter the earth's atmosphere as photons; namely, (1) they are quite uninfluenced by the earth's magnetic field, and (2) the ionization produced by them in a closed vessel does not increase continually in going to the top of the atmosphere, but passes through a maximum. It is shown to follow that the cosmic rays, in coming from their place of origin to the earth have not passed through an amount of matter that is appreciable in comparison with the thickness of the earth's atmosphere and that they must therefore originate in interstellar space rather than in the atmospheres of the stars. Some participation of the nucleus in the absorption of cosmic rays is brought to light.--Authors' abstract.

(182) ZUR METHODIK DER JONENZÄHLUNG IN DER FREIEN ATMOSPHÄRE

(CONCERNING THE METHOD OF COUNTING IONS IN THE FREE-AIR)

By Yo Itiwara

Physikalische Zeitschrift, Leipzig, vol. 32, No. 2, 1931, pp. 97-106.

The author points out that if ion tube-counters which are assigned for measuring the total number of all the small and large ions existing in the free air are used according to the method of charging (Auflademethode),--that is, the external electrode of the cylindrical condenser is kept at a constant potential and the charging of the inner electrode, which is connected with the earth, is measured,--not the total number of ions is measured, as has been accepted so far, but only one fraction of it.

The reason for this phenomenon consists of the fact that the charged external electrode produces at the place of the entrace of the ions a field by which a certain fraction of the ions is prevented from entrance into the counting apparatus.

The author proves by an experiment that large ions only are measured by the tube-counter described by A. Gockel and used by V. F. Hess during his measurements in Helgoland. Based on this fact, some results of Hess' measurements are to be corrected in order to obtain better conformity of his results with those procured by J. J. Nolan and his collaborators.--Author's abstract translated by W. Ayvazoglou.

The above article is followed by V. F. Hess's "Remarks concerning Yo Itiwara's work" in which Hess disagrees with some results of measurements obtained by Itiwara and recommends that Itiwara's experiments be repeated in the free air and under a different form.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(183) GEOPHYSICAL METHODS WITH SPECIAL REFERENCE TO COAL MINING

By E. Bein and H. Bertram Bateman

The Colliery Guardian, London, vol. 142, No. 3666, 1931, pp. 1182-1183.

In this article the authors point out in a general way the applicability of geophysics to coal mining.

They divide the problems which arise in respect to coal mining into two classes:

1. The direct location of coal seams and their position, thickness, etc.

2. The location of zones of disturbance associated with coal seams.

The applicability of the different geophysical methods is examined from the standpoint of exceptional cases, that is, cases where the circumstances are so favorable that the physical characteristics of a sufficiently thick seam make it definitely distinguishable from the strata above and below.

The following methods are discussed:

Gravimetric Investigations. - The torsion balance can not directly locate a coal deposit, but it can under certain circumstances locate the underlying stratum should the latter be inclined and show a distinct density difference from the overlying stratum. Topographical conditions should be as uniform as possible.

Electric Investigations. - The electrical method is inapplicable, as coal is a mineral which shows great variations of electric conductivity, depending on its composition and water content.

Seismic Investigations. - Exact depth estimates can be given only in cases where the order of the strata is such that the travel time of the elastic waves is always greater in the one stratum than in that overlying it, and less than in that of the underlying stratum. These conditions are sometimes found in brown-coal fields. Therefore, only in exceptional cases can the seismic method elucidate problems concerning the actual coal seam. Good results can be produced where it is a question of reaching the floor of the bed. Should the coal seam crop out, then there is every possibility of locating its strike, and even perhaps of determining its depth.

Geothermic Measurements. - Geothermic measurements can often be of great value and should be carried out systematically as drilling progresses.

Magnetic and Radioactive Methods. - There can be no question of using magnetic and radioactive methods for the direct investigation of coal deposits, ad the physical conditions necessary for the application thereof do not exist.

Of particular importance are zones of disturbance causing (1) displacement, and (2) decomposition of seams.

To the first class belong faults and thrust faults; to the second belong such occurrences of eruptive rocks, principally basalt and dolerite, as appear in some English coal fields, intrusions, sheets, etc.

It is quite certain that more may be expected from geophysics as a means of solving such problems than as a means of directly locating coal seams.
--W. Ayvazoglou.

(184) GEOPHYSICAL EXAMINATION OF METEOR CRATER

By J. J. Jakosky, C. H. Wilson and J. W. Daly

The Mining Journal, Phoenix, Ariz., vol. 14, No. 22, 1931.

Of the various theories advanced regarding the origin of Meteor Crater, two have been considered as most probable:

1. That the crater was formed by a meteroite or a swarm of meteoric material.

2. That it was formed by a "steam explosion" from hot solutions of gases.

The general purpose of the present geophysical work was to obtain information regarding the origin of the crater, and also as to the advisability of continuing with the exploration, and in particular where such exploration efforts should be directed.

Three main independent studies were carried out:

1. Geological. - The geology of Meteor Crater stratigraphy, its structure, conclusions on which the meteoric origin of the crater is based, as well as geologic age of Meteor Crater are discussed.

2. Electrical. - Factors governing electrical methods, apparatus used and general theory, plans of the electrical survey, and general summary of electrical work are examined.

3. Magnetic Investigations. - Theory, external field of meteoric mass, measurements, field procedure, masking effect of the crater-fill, regional gradient, anomalies in the crater, and the summary of the magnetic work are given.

In the "General summary of geophysical-geological results" the authors draw the following conclusions from the correlation of the geological, electrical, and magnetic studies:

1. From a geological examination the authors believe that the crater was formed by impact of a meteorite or a swarm of material.

2. The electrical survey gives indications of the presence of an area of higher conductivity in the southwest quadrant of the crater between the center and the rim. The main mass of this material lies at an effective depth of approximately 750 feet. A careful study of the original and altered materials found in the area indicates that this zone of higher conductivity is not due alone to fill material or structural conditions. The conclusions are that this zone contains meteoric material.

3. The magnetometer survey indicates the presence of an area containing magnetic material in the southern portion of the crater. This material starts at depths of approximately 200 feet and continues downward, probably concentrating with depth.

4. The geological evidence and the electrical and magnetic indications would individually be classed as fair or moderate effects of a buried mass. The general agreement as regards plan, location, depth, and other factors gives sufficient added strength to the results to warrant further explorations to the extent of churn drill holes, in the southwest gradient of the crater.

5. Results of the work are not considered sufficiently definite, in view of complicating factors encountered, to warrant calculations or predictions regarding the tonnage or mass of material which may be present. The chief result of the work has been to delineate definitely the area wherein future development work is to be concentrated.

6. The water level in the crater is comparable with the water level outside of the crater.

Figures of east-west and north-south sections, showing the probable structure of Meteor Crater, are given.--W. Ayvazoglou.

(185) GEOPHYSICAL SURVEYS IN OIL TERRITORIES

By J. B. Ostermeier

Oil News, London, vol. 28, No. 940, 1930, p. 546.

Speaking on geophysical methods of prospecting, the author states that gravimetric surveys in oil territories, particularly where the oil occurs in connection with salt deposits, have already reached considerable proportions by reason of the clear interpretation of the results of measurements allowed by this method. Seismic surveys give their most valuable results when comparative horizontal strata are involved.

These two methods are recommended for use with advantage when large areas have to be covered and if the preliminary work or its continuation is to be undertaken on a large scale and regardless of cost. The other methods discussed are the electrical and magnetic ones.

The possibility of the determination of synclines by geophysical methods and the particular importance of this fact to oil prospecting is mentioned.--W. Ayvazoglou.

(186) GEOPHYSICAL PROSPECTING DISCUSSED BY THE ST. LOUIS SECTION

By H. S. McQueen

Mining and Metallurgy, New York, vol. 12, No. 291, 1931, p. 172.

This paper is the report of a meeting held at the Missouri Athletic Association on January 17, 1931, at which H. A. Buehler, State Geologist, spoke on "The Results of Geophysical Prospecting by the Missouri Geological Survey During the Past Two Seasons."

The Geological Survey carried on field work in two branches of geophysics; namely, magnetometer survey and electric resistivity measurements. The studies have been confined to the iron ore deposits of the Ozark region, the lead deposits of southeast Missouri, and, to a lesser extent, to the high-alumina clay deposits of the north-central Ozark region.

The magnetometer surveys have been successful in outlining the configuration of the pre-Cambrian surface, and particularly the location of pre-Cambrian porphyry "highs" which in some instances bear a definite relation to the location of ore bodies in the adjacent sedimentary rocks. The results have also been applied to the solution of deep well water problems.

The Gish-Rooney method was used in the earth-resistivity measurements. This method of prospecting has been confined chiefly to the sinkhole type deposits of red hematite occurring in the central Ozark region. Detailed work indicates that the general area of the sink can be outlined and information obtained regarding the depth of the structure.

Measurements were also made over sink-hole type deposits containing diaspore clay. The thickness of the overburden and the thickness of the clay check very closely the results of drilling.--W. Ayvazoglou.

(187) THE GEOPHYSICAL DISCUSSIONS

Editorial note

Mining and Metallurgy, New York, vol. 12, No. 291, 1931, pp. 151-153.

This is a brief description of two sessions held by the Committee on Mining and Metallurgy on papers on geophysics. The following papers were discussed:

1. Resistivity Measurements upon Artificial Beds, by J. H. Swartz (see Geophys. Abs. No. 23).

2. Experimental Observations of Electromagnetic Absorption at the Mammoth Cave of Kentucky, a report on the results of work undertaken by J. Wallace Joyce, A. S. Eve, D. A. Keys, and F. W. Lee. Joyce discussed the relative value of various absorption formulas for different frequencies, and pointed out the necessity of applying experimental checks.

3. New development in electrical prospecting, by Hans Lundberg and Theodor Zuschlag (see Geophys. Abs. No. 25).

4. Electrical exploration applied to geological problems in civil enginerring, by E. G. Leonardon (see Geophys. Abs. No. 24).

5. Mathematical theory of electrical flow in stratified media, by D. O. Ehrenburg and R. J. Watson (see Geophys. Abs. No. 24).

6. Analysis of some seismic prospecting field data, by L. Don Leet and Maurice Ewing in which they presented some formulas for time distance curves and emphasized the importance of curved paths of seismic impulses.

7. A magnetic method of estimating the height of some buried magnetic bodies, by A. S. Eve (see Geophys. Abs. No. 24).

8. Practical geomagnetic exploration with the Hotchkiss Superdip, by Noel H. Stearn (see Geophys. Abs. No. 19).

9. Method for determining the magnetic susceptibility of core samples, by William M. Barret (see Geophys. Abs. No. 24).

10. Geophysical examination of Meteor Crater, Arizona, by J. J. Jakosky, C. H. Wilson and J. W. Daly (see Geophys. Abs. No. 25).

The meeting was adjourned after a strong request was voiced for more papers dealing with the intricacies of interpretation of the field readings, together with practical illustrations from experience of how conclusions are drawn from the data obtained. These are things that have not been divulged to any extent by commercial companies, who have left it to the Government experimenters to describe these secrets. For the benefit of all concerned, a more open-handed policy might well be adopted.--W. Ayvazoglou.

(188) OPENING OF COASTAL FIELD SHOWS VALUE OF GEOPHYSICS

By Jack Logan

The Oil Weekly, Houston, Tex., vol. 60, No. 8, 1931, p. 74.

The author enumerates a series of geophysically discovered salt-dome prospects in which production has been established with the first well following geophysical work (Sugar Land, Port Neches, and Bloomington fields in Texas, and Port Barre, East Hackberry, Lake Pelto, Bay St. Elaine, Dog Lake, and Cameron Meadows in South Louisiana).

Numberous other fields have come in with the second or third test following geophysical work, among them being Hankamer and Esperson; while there have been many other instances in which dome formation (cap rock or salt) has been established in one, two, or three tests.

Logan concludes by saying: "The fulfillment of the geophysical interpretation is noteworthy."--W. Ayvazoglou.

(189) GEOPHYSICAL PROSPECTING FOR OIL AND GAS

By James C. Templeton

The Petroleum Times, London, vol. 25, No. 630, 1931, pp. 175-176.

In this address delivered before the Oil Industries Club, London, Templeton outlined the most important methods of geophysical prospecting. He expressed the opinion that the gravitational and seismic methods were by far the most important ones, the magnetic method being used in areas where the geological conditions were particularly suitable for that method of investigation, while the different types of electrical methods had only a very limited application in oil prospecting. In connection with electrical methods Templeton made reference to the new development, namely the electrical logging of wells.

The great progress made in the improvement of geophysical instruments and equipment, as well as great advances in the mathematical analysis and the geological and geophysical relationship were discussed.--W. Ayvazoglou.

(190) ÉTUDE DE L'INFLUENCE DE QUELQUES FACTEURS GÉOPHYSIQUES SUR LES POINTS DE CHUTE DE LA FOUDRE

(STUDY OF THE INFLUENCE OF SOME GEOPHYSICAL FACTORS UPON THE POINTS WHERE LIGHTNING STRIKES)

By L. N. Bogoiavlensky

Laboratoire Radiologique de la Chambre Centrale des Poids et Mesures, Leningrad, U.S.S.R., 1931, 12 pp.

The author describes his studies on the conductivity of the atmospheric air and of the soil in localities often struck by lightning.

Four different methods were used:

1. Method of radiometric prospection according to penetrating radiation.

2. Measurements of the electrical conductivity of the atmospheric air by Gerdien's apparatus.

3. Method of electrometric prospection by using artificial electric fields.

4. Method of electrometric prospection of natural electric fields.

Conclusive results were obtained from the first three methods; the results have proved that the values of the intensity of terrestrial penetrating radiation, as well as those of the electrical conductivity of the atmospheric air are always higher in places often struck by lightning.

The electrical conductivity of the ground also reaches it maximum at the points with a maximum intensity of penetrating radiation.

The experiments were carried out in two different localities 200 kilometers from Leningrad, the ground of which consisted of sedimentary rocks (clay and sandstone) of the Devonian period covered with deposits in the form of bowlders of crystalline rocks of the ice period.

Thirteen diagrams are added to the article.--Author's abstract translated by W. Ayvazoglou.

8. GEOLOGY

(191) SHAPING THE EARTH

By William Bowie

Journal of the Washington Academy of Sciences, Baltimore, Md., vol. 21, No. 6, 1931, pp.

In this address delivered before the Academy in January, 1931, Bowie discussed the question of "Shaping the earth" under the following headings: (1) The crust of the earth; (2) formation of oceans and continents; (3) known facts about the earth; (4) isostasy; (5) the figure of the earth; (6) effect of topography on geodetic data; (7) variations of gravity; (8) comparison of Pratt and Airy hypotheses; (9) assumptions underlying isostatic investigations; (10) some isostatic conclusions; (11) isostatic adjustments and earthquakes; (12) objections to the contraction hypothesis; (13) diastrophic forces; (14) erosion and sedimentation; and (15) thermal changes in crust.

In his conclusion Bowie says:

"Isostasy is now widely recognized as a scientific problem. Its advocates hold that there is a maintenance of the isostatic equilibrium as materials are moved from one place to another over the earth's surface. These are the physical facts which are related to the processes involved in changes in the earth's surface. They have been proven by actual physical measurements. It has been stated that there are great horizontal movements in mountain areas, but that isostasy and its maintenance call for only vertical movements. My answer to this is that I recognize the horizontal movements in mountain areas, but believe that these horizontal movements are incidental to the vertical movements which are involved in maintaining the isostatic balance and which also result from the changes in the temperature of crustal matter brought about by the maintenance of equilibrium. There is an abundance of space in a mountain area for horizontal movements to occur, and it seems to me that it is easier to explain these movements as resulting from upward or downward moving material than as resulting from a shrinking interior of the earth and a collapsing crust.

Isostasy is a geological problem. It was outlined by the great geologist, C. E. Dutton. It has been used by the geodesists merely as an effective means by which to harmonize theoretical and observed values of geodetic data. The geodesists hope that isostasy may prove of great value to geologists in their effort to write the geological history of the earth." --W. Ayvazoglou.

(192) ISOSTASY FROM THE GEOLOGICAL POINT OF VIEW

By Rollin T. Chamberlin

The Journal of Geology, Chicago, Ill., vol. 39, No. 1, 1931, pp. 1-24.

Some of the outstanding geologic facts which have to be taken into account in a consideration of isostasy are enumerated. Mountain building, peneplanation, geosynclines, etc., are considered from the point of view of both geologic deductions and the isostasy theory. It is concluded that no single type of compensation or uniform depth of compensation meets the facts. The real test of the isostatic theory seems to lie in the limited regions of high relief. An attempt is made to show how geologic processes may operate to produce folded mountain ranges exhibiting the observed geologic phenomena and at the same time make it possible to satisfy the requirements of the geodetic determinations. Isostatic forces are in the whole equation of mountain-building forces, and play their appropriate part, but that part is secondary and subordinate.--Author's abstract.

9. NEW BOOKS

(193) Handbuch der Geophysik (Handbook of Geophysics). Vol. III, No. 1, 570 pp. 207 illus. "Kräfte in der Erdkruste" (Forces in the earth's crust), by B. Gutenberg; "Plutonismus und Vulkanismus" (Plutonism and Vulcanism), by F. von Wolff; "Erdkrustenbewegungen" (Movements of the earth crust), by A. Born; "Geothechnische Hypothesen" (Geotechnical hypotheses), by B. Gutenberg; "Mechanische Wirkungen von Eis auf die Erdkruste" (mechanical effects of ice upon the earth crust), by H. Hess. Gebrüder Borntrager, Berlin, 1930, Subscription price, 48 marks.

(194) Schlumberger, C. Étude sur la Prospection Électrique du Sous-sol (Study of the electrical prospecting of the subsoil). 2nd edition. Paris, Gauthier-Villars et C^ie^., Editeurs. 1930. 96 pp., 61 figs. Contents: Chapter 1. Trials of electrical prospecting before 1912; (2) electrical conductivity of various rocks and minerals; (3) theoretical study of the method for drawing maps of equipotential lines; (4) method of operation for the establishment of a map of equipotential lines; (5) application of the map to the study of stratified terrain; (6) study of the form of a conducting mass; (7) distribution of the current in depth; (8) incited polarization; (9) spontaneous polarization. The work of electrical prospection carried out in Sweden is briefly described in the annex.

(195) Seidl, Erich. Bruch und Fliess-Formen der technischen Mechanik und ihre Anwendung auf Geologie und Bergbau. Band II, Scher Formen; Band III, Zerreiss Formen (Break and flow forms of technical mechanics and their application to geology and mining. Vol. II, Shear forms; Vol III, Tear forms). V D I Verlag, Berlin, 1930, 22 pp. 51 text figs.; 81 pp. 145 textfigures. The following chapters are in preparation: I. Theoretical introduction; IV. Compression forms; V. Forms produced by bending; VI. Forms produced by flowage.

(196) Shaw, Capt. H., with the assistance of J. McG. Bruckshaw and S. T. Newing. Applied Geophysics. A brief survey of the development of apparatus and methods employed in the investigation of subterranean structural conditions and the location of mineral deposits. Contents: Introduction; magnetic method; gravity methods; catalogue of exhibits: magnetic, gravity, seismic, electrical; index. Ten plates with illustrations. London, published by His Majesty's Stationery Office, 1931. Price, 2 s. net.

(197) von Bernewitz, M. W. Handbook for Prospectors. McGraw-Hill Book Co., New York, 1931, 359 pp. Illus. $3. 2nd edition. The reason for issuing a second edition instead of another printing was to enable the author to add a 40-page chapter on prospecting by geophysical methods.

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3 - The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

I. C. 6501 August, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

ESSENTIAL FACTORS INFLUENCING SUBSIDENCE AND GROUND MOVEMENT

BY

W. R. Crane

I.C.6501,
August, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

ESSENTIAL FACTORS INFLUENCING SUBSIDENCE AND GROUND MOVEMENT[1]

By W. R. Crane[2]

INTRODUCTION

Subsidence and ground movement are phenomena common to and attendant upon practically all large-scale mining operations, and under certain conditions may occur in all workings, large and small.

The interpretation of the cause of failure in rock masses involves a study of the nature of planes of weakness common to all rocks. The dominating planes of weakness can readily be determined by observation in any given locality, and in order of importance in the study of subsidence are (1) faulting planes, (2) bedding planes, (3) joint planes, and (4) other prominent planes of weakness such as thrust and shear planes and contacts. This report has mainly to do with the third item, joint planes, and is a preliminary statement which is to be followed by a more complete report in bulletin form.

GENERAL CONSIDERATIONS

Planes of weakness make it possible for masses of rock to move more or less readily one upon the other and control the direction of movement. The purpose of this paper is to show the controlling factors under normal conditions. Abnormal conditions necessarily modify normal movements and present special problems.

In the absence of folding, faulting, and igneous intrusions, such as sills and dikes, the regular joint planes determine the extent and direction of ground movement. In the following pages conditions affecting the mechanics of ground movement are explained.

This report is based wholly upon facts obtained by observation on actual failure and movement of rock masses in mines and opencuts, involving 150,000 instrument readings on the strike and dip of joints and slips.

METHOD OF PROCEDURE

The method of procedure in making the observations recorded in this report, together with their classification and interpretation, are discussed in detail.

Observations on the strike, and on the angle and direction of joints and slips were taken on rock exposures either on the surface or underground, or in both places, for distances of one hundred feet or more and at intervals of several hundred feet, the areas tested being well distributed over the property or locality studied.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6501."

2 - Senior mining engineer, U. S. Bureau of Mines.

The data collected were classified according to angles of bearing and dip, the number of observations taken on certain groups of angles being determined by their relative prominence. The relative prominence of the angles thus measured forms the basis for determining the angle upon which failure will take place, or the so-called angle of draw.

Of the information secured from the observations, that of the dip of the joint planes is most important; next in importance is the bearing or strike of the joint planes, as the application of the angle of draw to the periphery of the workings depends upon it.

CONTROLLING FACTORS

The determination and recording of systematic planes of weakness that are controlling factors in the failure and movement of rock masses form the basis for determining the direction of movement when failure takes place. Rocks break according to system unless disturbed by irregularities largely due to cross-bedding in stratified formations, schistosity in metamorphic rocks, and flow or thrust planes in igneous rocks. The series of joints in any particular locality must be determined by observations on their strike and dip, which define planes along which the rock fails most readily and in accordance with the prominence of the joints determine the planes of movement of rock masses and the angles of draw.

The terms "joint" and "slip" are commonly used by miners in referring to the same phenomena, but they are not identical in definition. A joint is a plane or slightly curved crack which is one of an approximately parallel set ranging from a few inches to many feet apart. Slips are cracks along which movement of the rock has taken place. For the purpose of this report the term "joint" will be used to cover both joints and minor slips in which little movement has taken place.

Joints and slips occur in all rock formations and are arranged in series made up of pairs, at right angles or nearly so. They may be quite rough, particularly in certain rocks such as coarse-grained sandstones, and, depending somewhat upon the direction taken across the formation, one of the planes of a set may be smooth and the other rough.

Shearing is that deformation of rocks brought about by accumulative small lateral movements along innumerable parallel planes, generally resulting from pressure due to folding, faulting, and igneous intrusions, all of which produce schistosity, cleavage, and other metamorphic structures. Shear zones are those in which shearing has occurred on a large scale so that the rock is crushed and brecciated.

Fissures are extensive cracks, breaks, or fractures in rock masses. They differ from joints and slips in width, the latter being generally closed and very regular. Fissures are usually prominent planes of weakness and must be given due consideration in a study of ground movement.

Other planes of weakness that are important factors in subsidence and ground movement are bedding, faults, contacts between formations including dikes and sills, and unconsolidated formations. Rock masses may be extensively broken by earth movement, the rock being shattered into small pieces with no semblance of order or system. Such shattered areas may be considered as unconsolidated formations so far as ground movement and subsidence are concerned.

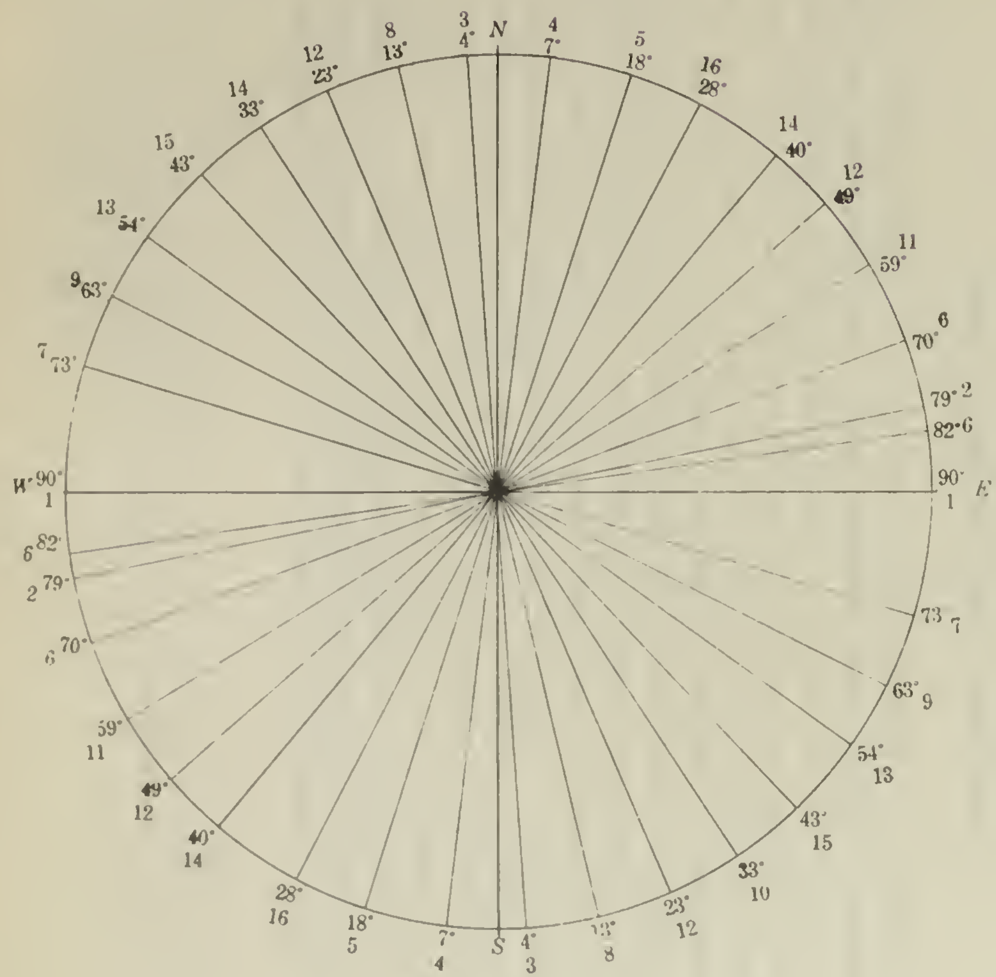

Figure 1.—Relative prominence and bearings of joints in a typical case

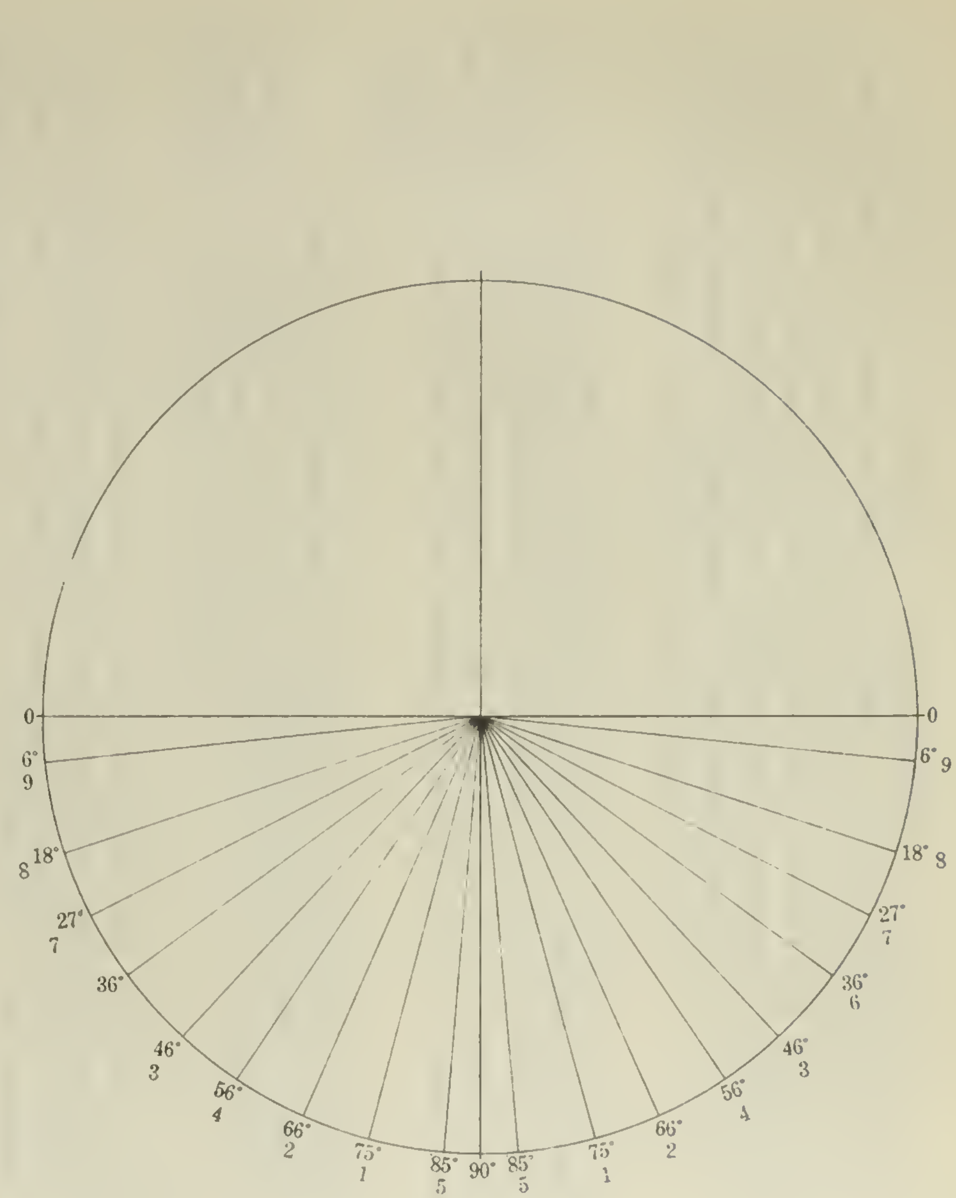

Figure 2.—Relative prominence and vertical distribution of angles of dip of joints in a case typical for both igneous and metamorphic rocks

Draw, strictly speaking, is the distance on the surface to which the subsidence extends beyond the caved workings. The angle of draw is the angle formed with the horizontal by a line connecting the outer edge of the draw on the surface with the nearest point of the caved workings. The angle of draw may not be permanent but may decrease with time or as workings are extended laterally or in depth. When the force of gravity exceeds the frictional resistance between the portions of rock affected, movement occurs on a plane extending from the workings to the surface along the angle of draw.

All rock formations regardless of kind or condition are cut by series of joints, each series representing a movement of the formation. The simplest condition is represented by a pair of joints at approximately right angles; the more complex may have a score or more pairs of joints. When rock formations are broken due to the removal of underground support, the movement occurs upon existing planes of weakness and not upon fresh breaks across the formations.

A method of systematically recording the essential data concerning joints is indicated by the arrangement of bearings and angles of dip as shown in Figure 1 and 2. Relative prominence is based upon the number of joints and slips observed and to a less extent upon the size of the individual faces exposed.

Normally, bedding planes are the most important factors in failure and movement of rocks; next are faults and contacts, while jointing, shearing and schistosity are less important. However, the more general occurrence of joints places them in a position of great importance as the controlling factor in subsidence and ground movement when bedding planes and faults are absent.[3]

CAUSE AND NATURE OF MINE SUBSIDENCE

With adequate support in mine workings no subsidence or movement will occur in the superincumbent and inclosing rocks. When movement takes place it continues until the openings are closed and equilibrium is established. The direct and visible result of the readjustment of the adjacent rock is fracturing, which may eventually reach the surface. The size and depth of workings are important factors in determining the extent of the surface disturbances. Other factors influencing the mode of failure in subsidence and ground movement are shape and position of ore body, character and condition of ore and associated rocks, and the method of mining employed. With workings in small ore bodies and particularly narrow ones, failure of overlying rocks rarely ever reaches the surface except where the cover is shallow. Under deep cover failure may or may not take place. Where conditions are favorable for failure of rock masses overlying and adjacent to workings, breaking occurs progressively under some conditions, suddenly under others, and proceeds from the top of the workings upward to the surface and gradually spreads laterally to the ultimate angle of draw. The resulting phenomena are fracturing, settling and caving of the surface. Observations covering a period of years and extending over many mining sections have proved that rocks break in an orderly and systematic manner and not in an irregular or haphazard fashion, as is commonly supposed.[4]

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3 - Crane, W. R., Subsidence and Ground Movement in the Copper and Iron Mines of tho Upper Peninsula Michigan: Bull. 295, Bureau of Minos, 1929, p. 5.

4 - Crane, W. R., Work citod, pp. 5 and 6.

Bedding Planes

Bedding planes are the planes or surfaces separating the individual beds of a sedimentary rock. They are natural planes of weakness in rock masses, regardless of whether they are bonded or not; furthermore, the irregularity of their distribution or of the spacing between the respective layers or strata of a formation contributes to irregularity in subsidence and ground movement.

Bedding planes may be smooth or they may be very rough, irregular, and undulating. Even with slight tilting or folding they are weakened or opened, thus permitting water to enter and still further weaken them through disintegration of the rock. Irregularities of formation, such as rolls and cross-bedding and particularly the latter, are disturbing factors in ground movement.

Flow surfaces in igneous rocks are often very pronounced and resemble bedding planes; however, they are usually rough and markedly irregular in strike and dip. At angles under 30° they can not be considered of importance as controlling factors in ground movement. It is only when bedding planes are inclined toward workings that they facilitate the movement of rock masses. When dipping away from workings the effect of bedding planes is practically negative. Therefore, it is only in exceptional cases that bedding need be given serious consideration in its effect upon subsidence and ground movement (see fig. 3).

Nature of Joints

Joints are of universal occurrence in rocks and therefore must be considered in connection with ground movement under all conditions. They vary in prominence from a mere plane of weakness to definite breaks often taking the form of open fissures and containing considerable quantities of water. The joints that occur under normal conditions are those to which the author has given particular attention. The larger ones, called "crossings" by the author, are abnormal as to their size and influence on ground movement but are numerous in certain localities and their occurrence must be given special consideration as in the case of faults.

Joints as a rule are not only universal in occurrence, but are extensive, that is, they are very persistent in bearing and dip, and whereas an individual plane may definitely terminate, the plane of weakness continues indefinitely and maintains a definite bearing until deviation is produced by movement and displacement of the formations by folding, by igneous intrusions, or by some action subsequent to the formation of the joints. The study of joints has shown that they differ in character according to their mode of origin. The usual and normal type, and by far the most important, may be designated as regular joints; their origin is not definitely known, but it is evident that a phenomenon of such universal occurrence must be produced by a powerful force. Other joints may be said to have known origins, the terms employed in their designation usually indicating the character of the force responsible for their production - as for instance shear, thrust and flow planes. Curved planes are produced by cooling of igneous rocks, although the more extensive ones may have resulted from stresses producing thrusts. None of the planes of weakness designated as joints are extensive except the regular ones; all others may, therefore, be considered as local phenomena, their beginning and ending often being observable.

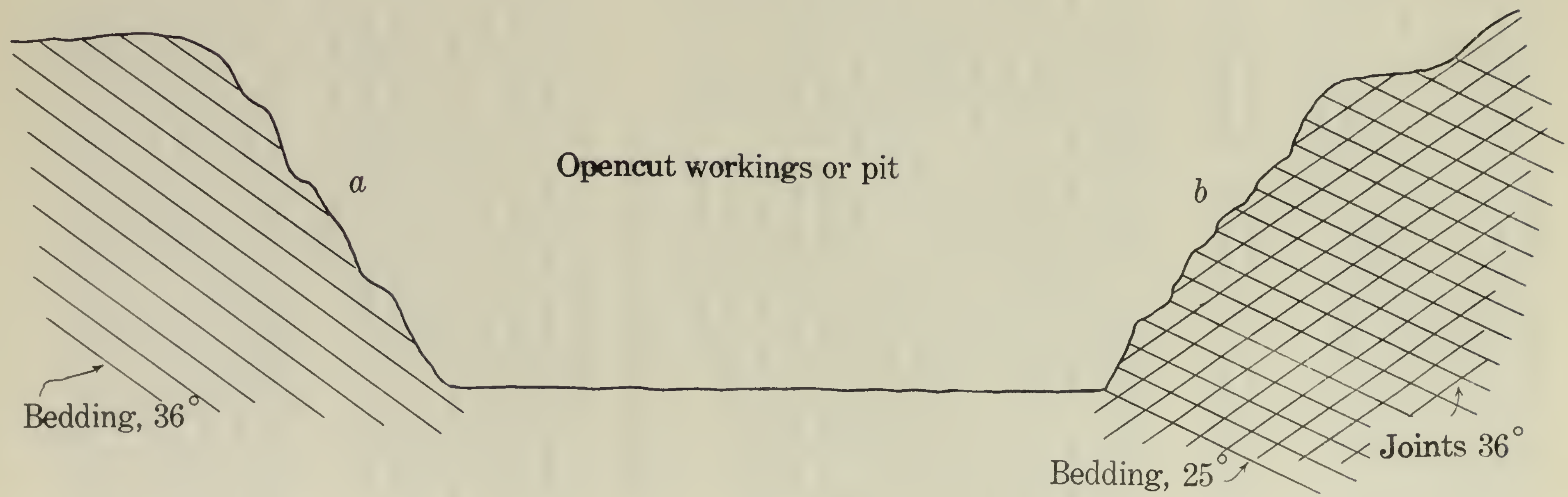

Figure 3.–Vertical section through opencut workings, showing arrangement of bedding and joint planes. Bedding is usually more prominent than joints, and controls ground movement if the beds dip toward the workings; otherwise they have little effect and joints or slips become the controlling factors. On the *a* side of workings bedding controls ground movement, on the *b* side the joints control, therefore the angle of draw is similar on both sides, as the angles of bedding on one side and of joints on the other are the same. This is an actual case, although not drawn to scale

The time element is an important factor not only in the formation of joints but in their prominence. Unconsolidated formations do not usually take on regular jointing until through pressure and cementation they become hard and strong enough to retain a joint. The character of the rock has an important bearing upon the formation of joints and under especially favorable conditions a relatively short period of years may be sufficient to form them. Much remains to be learned regarding the origin of joints, but in a general way it appears from extended observations that fine-grained and thinly stratified formations are most susceptible to their production.

A typical arrangement of joints, the result of a large number of observations at one of the localities studied by the author is shown in Table 1, the prominence being indicated by numbers.

Other and more irregular planes, such as shear, thrust, and flow planes, and dip slips, also occur in rock masses with the regular joints and often complicate an otherwise simple interpretation of the factors affecting ground movement. Moreover, certain arrangements of these irregular planes, may result in their superceding the regular joints in importance as controlling factors in ground movement. Dip slips, which are planes paralleling the dip of the general structural features, occur in all formations alike, igneous and metamorphic as well as sedimentary, and have a much wider application to problems of subsidence than bedding. When they occur in stratified rocks their principal effect is to weaken the bedding planes, permitting movement to take place more readily upon them. In unstratified formations they produce an effect similar in all respects to that produced by bedding. The distribution of strike of joints is shown in Table 1 the most prominent angles being those in column P-1. Curved slips have slight importance as factors contributing to failure and movement in rock formations. They are very local and while prominent are of local application only. Radial joints in folded formations are likewise too irregular and lacking in system to be considered in any study of ground movement.

Occurrence and Nature of Faults

Faults are definite breaks in the continuity of rock formations attended by a movement on one side or the other of the break along the plane of the fault. In this paper we are concerned with four classes of faults; normal or downthrow, thrust or upthrow, rotary or scissor, and curved faults. The normal and thrust faults are most common, while the rotary and curved forms are comparatively rare. The different forms of faults are shown in Figure 4: in A the direction and dip of joints are not affected by the faults; in B the dip of joints is changed, being reduced by a downthrow; in C both strike and dip of joints are changed, the amount depending upon the degree of rotation. D shows how an intrusive may change the dip of bedding and joint planes.

The effect of faults upon ground movement is the same as that of bedding, large joints or crossings, and in fact of all prominent planes of weakness; that is, if the fault plane is inclined toward the workings, movement takes place upon it. On the other hand, if a fault dips away from the workings, movement in the rock occurs not along the fault but across it, following some other plane of weakness. With rotary or scissor faults, planes of movement are rendered discontinuous by the displacement of the bedding or slips which may cross one another. Joints formed after faulting and rotation may present new planes of weakness upon which movement may act but such planes will be nearly vertical.

Table 1.- Distribution of Strike of Joints
(P indicates order of prominence, A is the average angle)

0 - 10°		11 - 20°		21 - 30°		31 - 40°		41 - 50°		51 - 60°		61 - 70°		71 - 80°		81 - 90°	
E	W	E	W	E	W	E	W	E	W	E	W	E	W	E	W	E	W
2	8	18	14	29	24	31	39	50	48	53	60	67	67	77	74	84	87
10	8	20	16	28	29	33	38	47	48	51	53	68	65	73	78	86	83
6	A-8	12	20	28	26	36	38	48	46	53	57	70	65	71	73	82	88
A-6	P-15	11	15	24	21	38	40	49	42	52	52	70	70	73	77	88	83
P-14		15	A-18	30	30	35	36	42	44	54	51	68	63	77	72	87	82
		A-15	P-13	28	27	36	35	47	47	51	55	62	64	77	A-75	84	85
		P-12		28	27	36	34	48	45	58	52	66	A-66	72	P-12	89	82
				25	30	35	40	45	42	54	60	62	P-11	78		82	84
				25	24	37	35	48	50	54	A-55	62		72		82	A-84
				30	23	38	A-37	50	50	54	P-10	66		72		83	P-10
				30	24	34	P-9	43	49	60		63		76		A-85	
				27	21	39		50	A-46	56		63		74		P-8	
				29	A-26	32		45	P-7	52		70		80			
				21	P-6	32		42		57		62		80			
				30		31		47		57		64		80			
				29		37		45		51		68		80			
				30		34		44		52		65		76			
				30		36		42		52		A-66		A-76			
				28		32		46		57		P-5		P-5			
				28		38		50		59							
				23		33		46		51							
				30		40		42		A-54							
				21		40		46		P-4							
				26		34		41									
				30		32		46									
				21		34		45									
				30		35		49									
				28		37		42									
				30		38		49									
				28		32		49									
				30		40		43									
				25		36		A-46									
				A-27		37		P-3									
				P-2		37											
						32											
						32											
						37											
						36											
						32											
						36											
						33											
						35											
						A-35											
						P-1											

Faults may intersect ore bodies and workings within them, and may or may not come to the surface. Without full information relative to the position and extent of faults, it is impossible to foretell the extent of ground movement that may occur.

Contacts Between Formations

The contacts between stratified rocks may be unconformable, whereas contacts between igneous and sedimentary rocks are unconformable except in the case of lava flows and tuffaceous beds which may form a part of a sedimentary series. Owing to the frequency in occurrence of igneous instrusions, particularly in the large metal mines of the west, contacts are numerous and may or may not be important in their influence on ground movement.

Intrusions naturally follow along lines of structural weakness in the invaded rock. When an intrusion of igneous rock follows bedding planes it is called a sill, even if it shifts back and forth between adjacent beds. When a long incursion of igneous rock is made across a bedded formation, it becomes a dike. Dikes cut across beds, and if the contacts are relatively weak they may have a controlling influence on rock movement. The disturbance and weakening of formations by the intrusion of igneous rocks promotes conditions favorable to the breaking down and movement of rock masses.

Schistosity

Schistosity is often developed in metamorphic rocks and is quite variable in character, ranging from an evenness and regularity resembling stratification to a gnarled condition which at points may be angular. There is a marked difference in other characteristics between the gnarled and crumpled areas and the long and smoothly sweeping curves of schistose formations, the former appearing to be stronger and breaking on joints rather than parallel with the schistosity. Failure of a rock mass, however, is facilitated by schistosity, movement taking place quite readily upon it, which with the occurrence of numerous joints cause the rock to fracture and break up readily under ground movement.

Occurrence and Nature of Shatter Zones

Shatter zones are belts in which the rock is cracked in all directions resulting in a network of small fractures. The distribution of shattered areas usually is coextensive with the intrusive areas of igneous rocks; they usually occur along fault planes, and although they occur on both hanging wall and footwall sides they favor the latter. In areas where faults are numerous, shatter zones may occur at more or less regular intervals and may be several hundred feet wide and extend from hundreds to a thousand feet or more vertically.

Such areas of badly broken rock have a profound influence on the failure and movement of rock masses, and as the former and regular planes of weakness have been obliterated, the action accompanying failure of rocks may be very irregular and may afford no definite means of determining either extent of failure or direction of movement. However, as a rule, shattered areas are consolidated to such an extent that incipient planes of weakness or joints are present, thus controlling their failure in a more or less definite way. The breaks as observed by the writer in such formations are vertical or nearly so, thus providing a simple method of determining extent and direction of failure.

Character of Rock

There are two general types of formations that need particular consideration in connection with subsidence and ground movement - namely, consolidated and unconsolidated rocks. The latter are composed of sands, gravel, and cobble stones often associated with clay; the former are solid rocks, stratified, metamorphic and igneous. Any rock of recent or comparatively recent origin that has not become relatively hard and strong by pressure and mineral bonding, thus involving the time element, may be classed as unconsolidated. Such rocks are not dense and firm enough to have become jointed; and breaks in such formations range from the vertical to the angle of repose as limits.[5]

Semiconsolidated formations, on the other hand, have become jointed; and although the joints may not be prominent, they are usually vertical, provided the formation has not been disturbed by earth movement. Typical semiconsolidated formations comprising by far the larger number of this class, are conglomerates formed in river valleys and lake beds. Recent beds of fine sand and silt deposits may also be designated as semiconsolidated if joints are present.

A wide range of rocks with variable characteristics belong to the consolidated class. These rocks are those most commonly encountered in mining and consequently are of particular interest in connection with failure and movement in rock formations. Considering certain characteristics only, the hard, dense rocks are more susceptible to the formation of joints. Softer and tougher varieties of rock often resist systematic fracturing to a marked degree; however, planes of weakness paralleling the joint planes may exist although invisible. As prominence of joints means easy breaking and movement in rock masses, the harder the rock the more readily is ground movement promoted. Thin, hard beds are particularly susceptible to the formation of joints and consequently to failure.

Summary

The cause of failure and movement of rock masses is not confined to artificial excavations, as is evident from numerous occurrence of rock slides on rough and precipitous ground. Although any weakening of support is responsible for the setting up of stresses that ultimately cause failure and movement, the main and essential difference between natural and artificial excavation is the time involved in producing these effects, natural agencies being as a rule relatively slow excavators, whereas the systematic efforts of man are locally very rapid.

Excavations are the immediate cause of failure of rocks. The controlling factors are existing planes of weakness; and the results are breaking down or caving within the excavation, settlement or subsidence in superincumbent rocks, and movement in rocks adjacent to the excavation.

The controlling factors are bedding or stratification, faults, contacts between formations, and joints or slips. All of the factors mentioned except joints may be classed as special, while joints are universal in their occurrence and application. The essential characteristics of joints that give them importance as contributing and controlling factors

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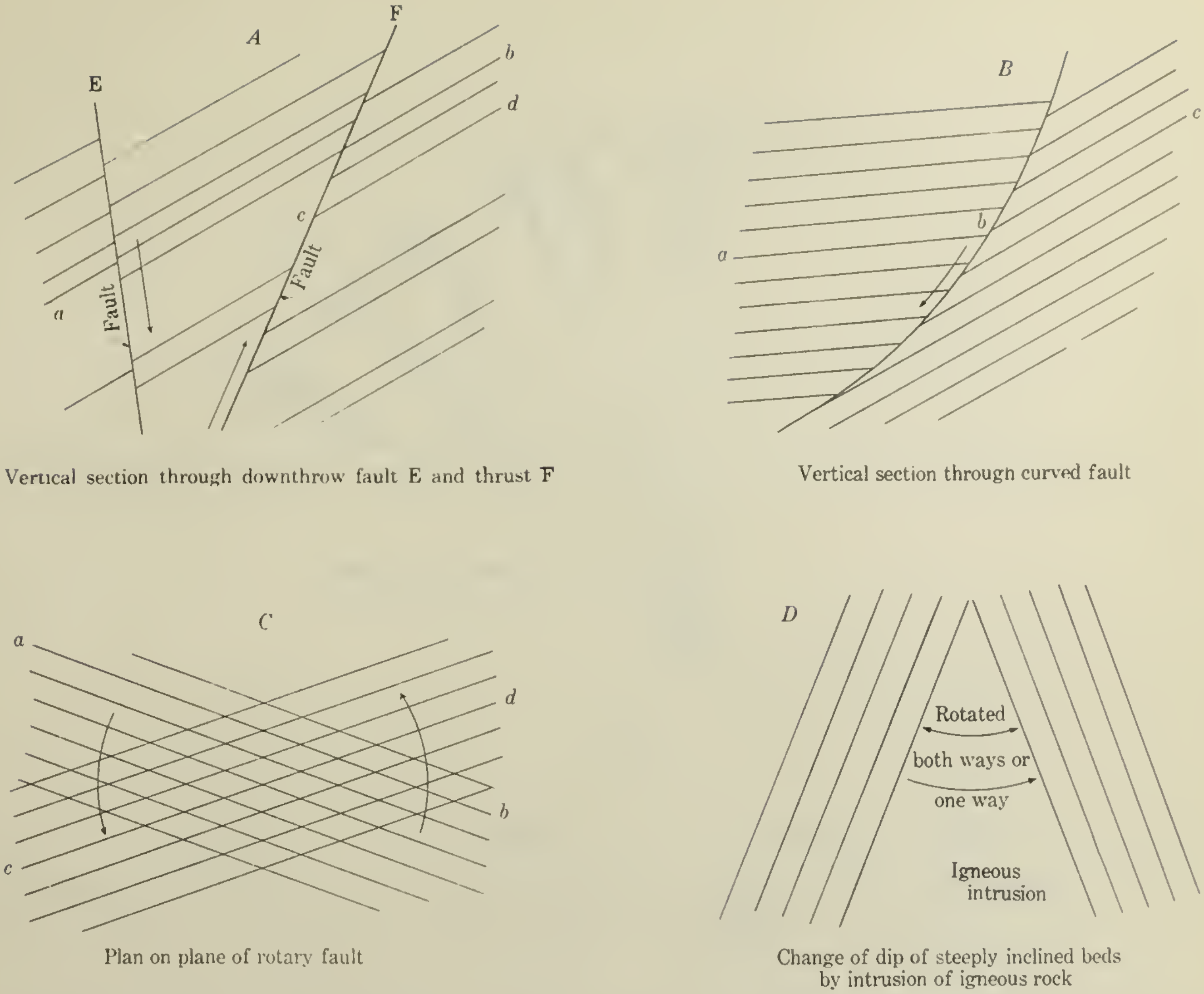
5 - W. R. Crane, Subsidence and Ground Movement in the Copper and Iron Mines of the Upper Peninsula Michigan: Bull. 295, Bureau of Mines, 1929, p. 34.



Vertical section through downthrow fault E and thrust F

Vertical section through curved fault

Plan on plane of rotary fault

Change of dip of steeply inclined beds by intrusion of igneous rock

Figure 4.—Sketches showing effect of faults in changing strike and dip of bedding or joint planes

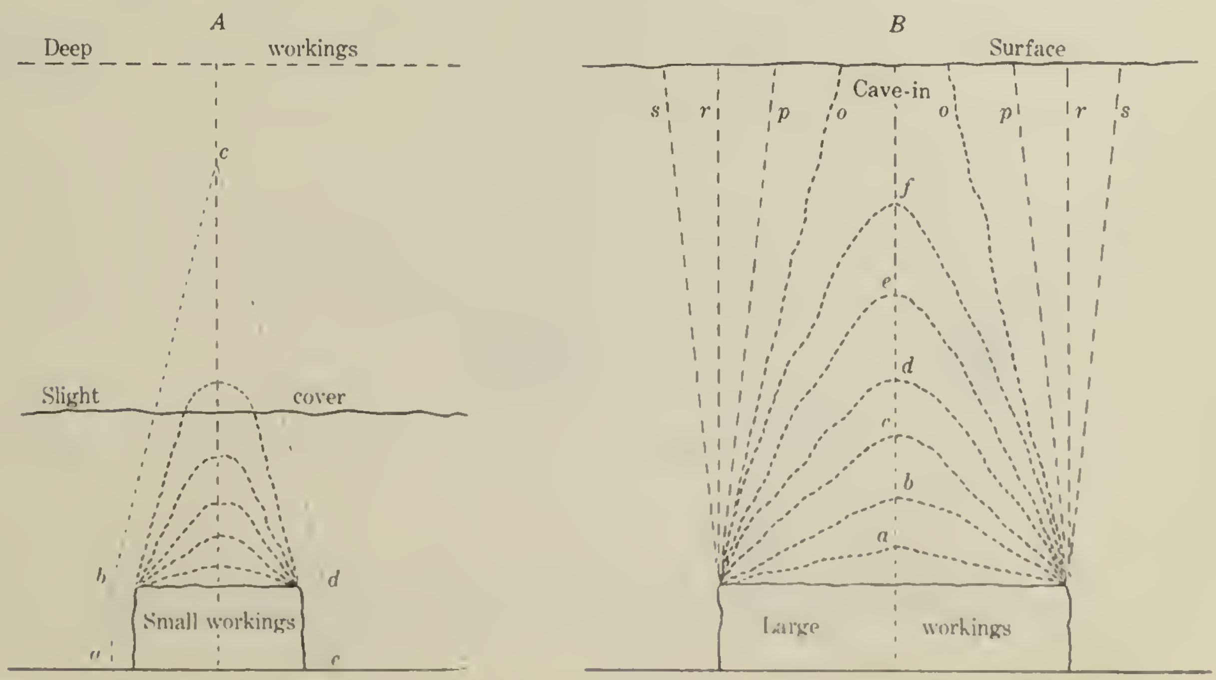


Figure 5.—Vertical section, showing how failure of roof of workings affects overlying rock

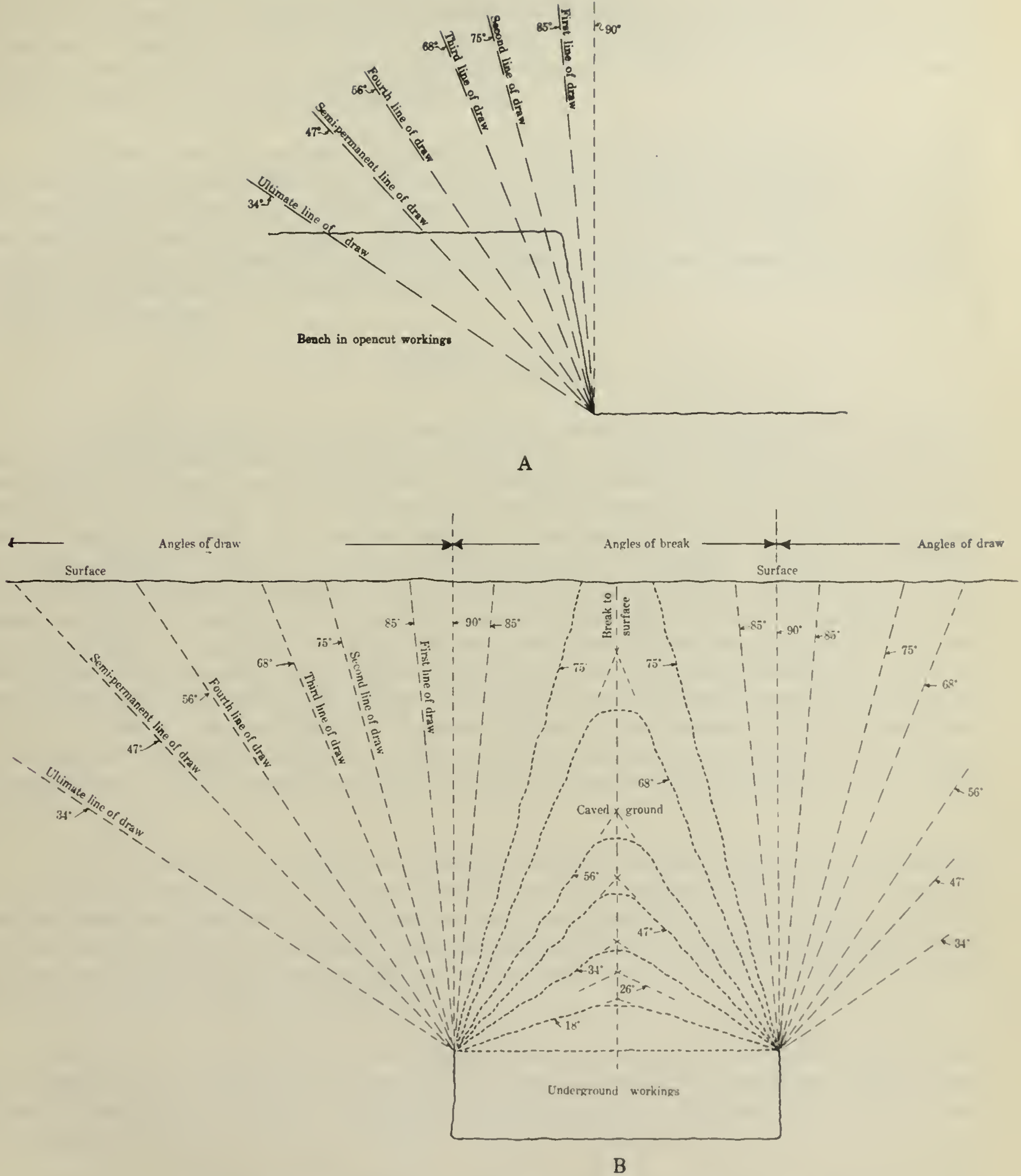


Figure 6.—Progressive decrease in the angle of draw in bench A and underground workings B

in subsidence and ground movement are: large extent both horizontally and vertically, systematic arrangement, and relative smoothness of surface. Other factors such as bedding, faults, contacts, etc., may be much more prominent than joints locally, but aside from bedding are irregular in occurrence. Bedding, while prominent in sedimentary formations, is confined to them and may or may not be more prominent than joints. Furthermore, like all other planes of weakness, bedding must be in the proper position to be an effective agent in ground movement and therefore is of limited application only.

Extended observations under all mining conditions demonstrate beyond question that failure and movement take place upon systematically arranged planes of weakness or joints in rock masses, which constitute the most important factor in failure of rock overlying underground workings Caving in stopes, natural stoping, breaks to the surface or cave-ins, settlement of surface or subsidence, and ground movement all present indubitable evidence of the effect of joints as an ever present factor in contributing to and controlling failure and movement in rock masses.

MODE OF FAILURE

The breaking down of the roof and walls in underground workings and the walls of opencuts is due to the pressure of the overlying rocks. The extent of failure is determined by the size of workings, particularly their horizontal dimensions, and by the character and physical condition of the rock. The height to which failure will reach depends upon the width of opening or stope, provided the length is considerably greater than the width, and unless the thickness of cover is slight and breaks down, the height will remain constant until the width of the opening is increased by failure of the walls. Natural stoping increases the width of opening and in turn increases the height of failure in proportion to the increase in width of opening. Figure 5A shows small and narrow workings which will not break to the surface unless the cover is slight. In large workings, failure of roof continues upward until the surface is reached, as in Figure 5B, passing successively through a, b, c, d, e, f and o, the last stage, o, breaking through to the surface. From this point on, failure extends laterally through p, r, and s to the ultimate draw. The walls of workings are extended by natural stoping permitting caving to a greater height (See a, b, c, d, e in fig. 5A).

It is evident that there is a definite relation between depth of cover and the width of workings necessary to bring about failure of the surface, but owing to differences in the character and condition of the rock no universally definite ratio exists. Furthermore, the excessive weight of adjacent rock masses, presence of water in the rock and other conditions affecting and influencing failure and movement of rock, may be factors seriously affecting the normal procedure in failure.

A cave-in having resulted from breaking down of the roof of a stope, failure continues upward to the surface more or less rapidly until the walls are vertical. Then breaking down of the walls commences. The action now becomes more a sliding than a falling one and is designated as draw, as gravity pulls or draws the rock into the pit.

There is a progressive extension of draw from the vertical to the ultimate angle of draw. This is illustrated in Figure 6 for both bench and underground workings. In the latter a distinction must be made between failure through breaking of roof and that through breaking and draw in the adjacent rock, which is shown in Figure 6B, although the same lines

of weakness control in both instances. The first movement or draw is on the steeper angles and is usually quickly accomplished, gradually slowing up as the angles become flatter until further movement is impossible because of friction. The failure in benches of opencuts is similar to that of underground workings, but less pronounced.

Faults and dikes may intersect ore bodies and the inclosing formations, modifying the effect of jointing.

The essential points relative to failure of rock masses may be summarized as follows: (1) Rock breaks according to a systematic arrangement of planes of weakness, with slight irregularities due to breaking between joints; (2) measure of the strike and dip of the joint system is essential; (3) the strike and dip of the most prominent joint planes indicate the direction of movement, the dip being the most important as it gives the angles of draw; (4) the angles of draw show the ultimate limits of breaking or subsidence.

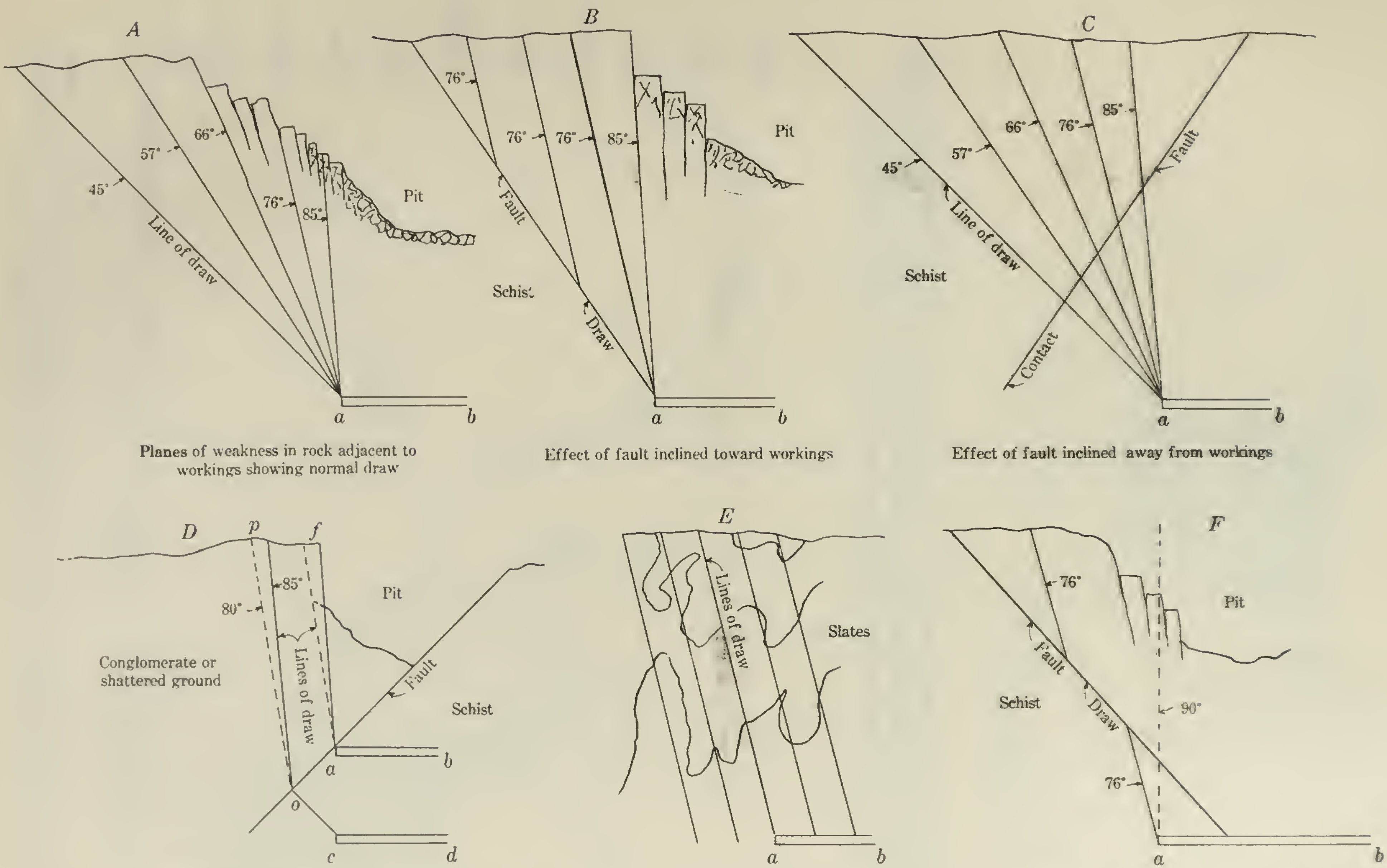
Figures 7, 8, and 9 illustrates various aspects of subsidence and ground movement ranging from movement resulting from normal conditions of draw to the more complicated action resulting from the presence of faults.

DETERMINATION OF STRIKE, DIP, AND DIRECTION OF DIP OF JOINTS

The strike of joints must be known to determine the extent and continuity of the planes of weakness in a given locality or district. The dip of joints, however, has a more important bearing on subsidence and ground movement; the direction of dip is secondary only to the angle of dip, as it gives the direction of movement of rock masses.

Observations on joints should be taken first in a section where they are regular. If a disturbed section is encountered either omit observations therein or make careful selection of joints so that undue importance may not be given to particular joints of the series. For instance, a large number of parallel joints may occur in a sheer zone. Observation taken on all such joints would obviously introduce an error, since their prominence would be exaggerated for the locality as a whole. It is needless to say that care must be exercised in taking observations, as it is upon them that the accuracy of results depends. However, experience in areas of regular jointing readily prepares one for the more difficult task of making proper selection in disturbed localities.

Observations are grouped according to bearing and dip, the groupings indicating the relative prominence of certain joints in the series for a locality. (See Table 1 and fig. 10.) The ideal grouping for a quadrant or 90° would be in 90 columns; then the number of observations in any given column would represent the relative prominence of a certain angle. However, this arrangement is too cumbersome and involves unnecessary refinement. Obviously the error in selecting angles increases with the reduction in number of columns employed in the grouping, and consequently there is a lower limit in the number of columns which should be used. The smallest and at the same time the most convenient number of columns is nine, with ten degrees to the column. While the spread of angles in a column is theoretically ten degrees, yet in reality it is usually limited to about six degrees. In the spread of angles the true angle is just as likely to be below as above the average but the average angle is probably the fairest approximation that can be made.



Planes of weakness in rock adjacent to workings showing normal draw

Effect of fault inclined toward workings

Effect of fault inclined away from workings

Effect of double draw with workings at two levels

Uniform draw in folded formations

Combined draw on fault and slip below fault

Figure 7.–Vertical sections showing effect of faults and joints on rock movement. *A*, Normal condition; *B*, broken ground moving a fault toward the workings; *C*, lack of effect of a fault or contact inclined away from the workings; *D*, effect on draw of two different formations cut by fault with workings at different levels; *E*, joints cutting folded formations in regular and persistent manner; *F*, a double draw on a fault and joint planes

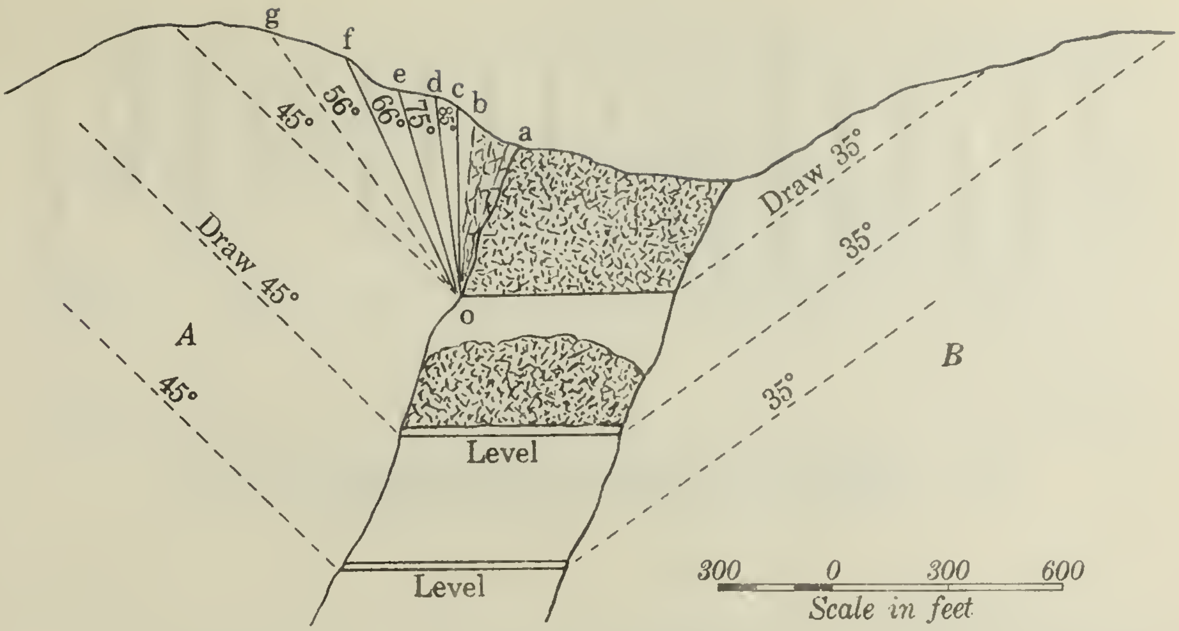


Figure 8.–Vertical transverse section through ore body with different kinds of rock in hanging walls and footwalls at *A* and *B*. The angles of draw are 45° in *A* and 35° in *B*. Pronounced failure has occured at the vertical co, while draw has extended back as far as fo or to an angle of 66°

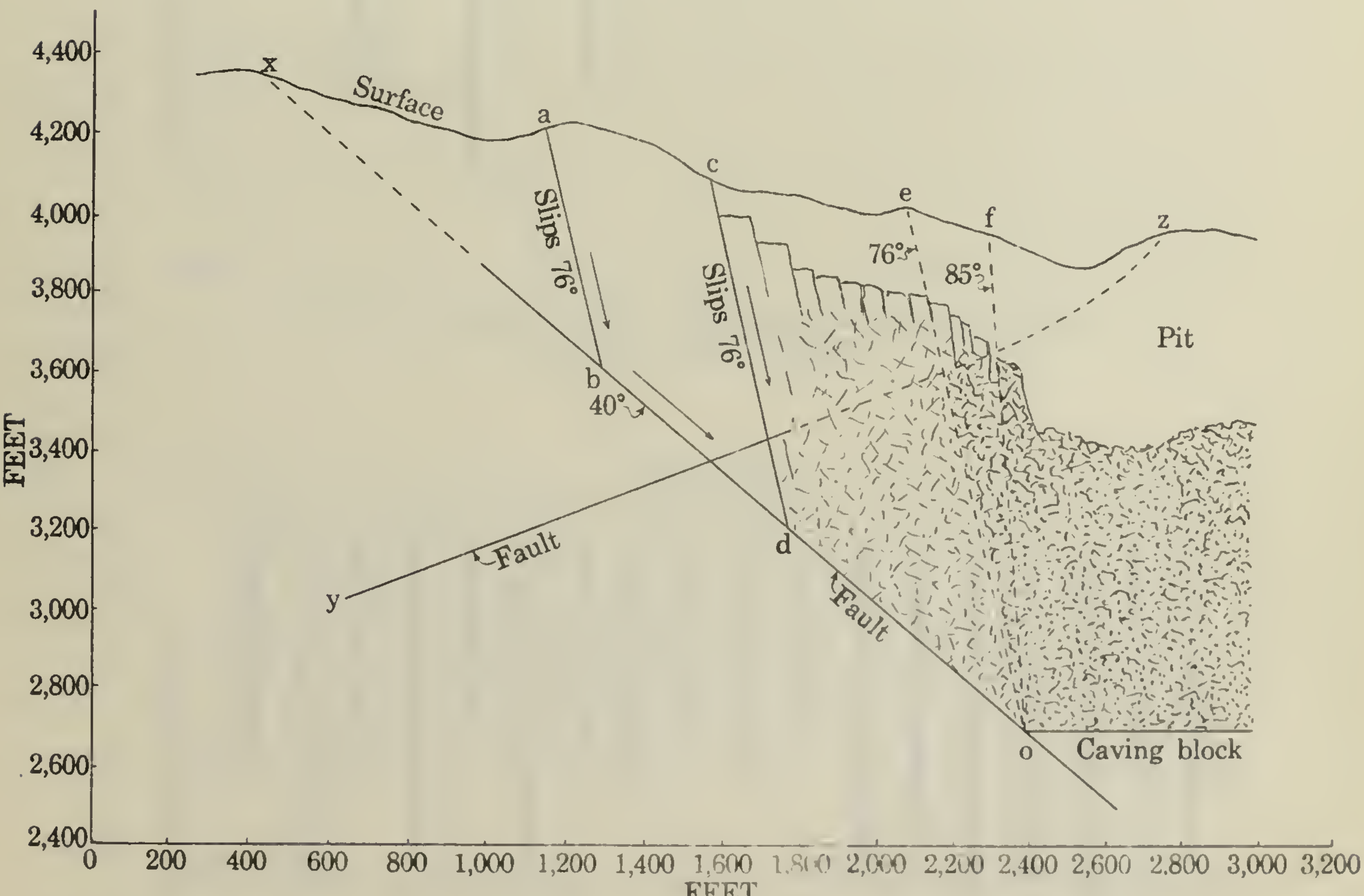


Figure 9.–Vertical section through caving block showing draw taking place on fault xo and cutting across fault yz, the former controlling movement, the latter having no effect upon it. Breaking and failure is taking place on slips above fault plane as shown at ab, cd, and at e and f

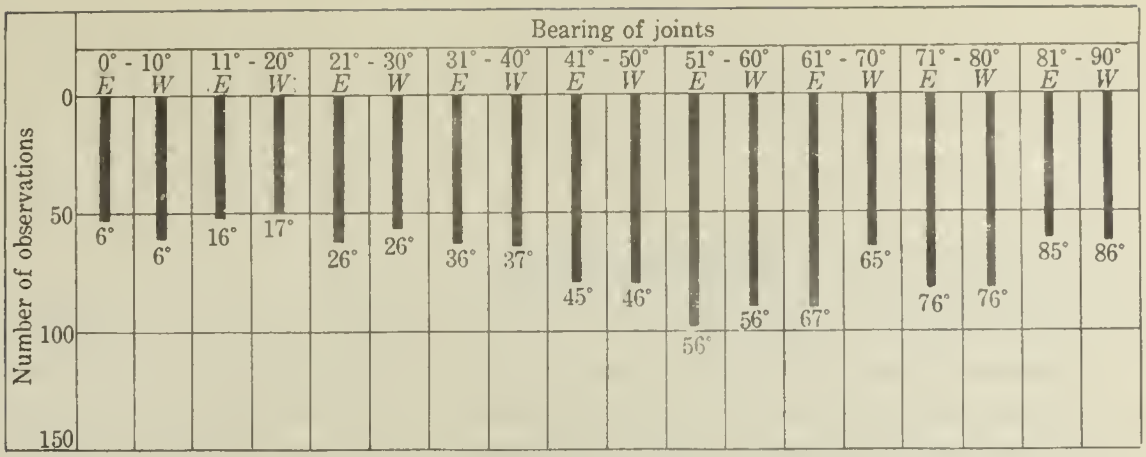


Figure 10.—Distribution and prominence of bearings of joints in a given locality, indicating equal prominence in all directions

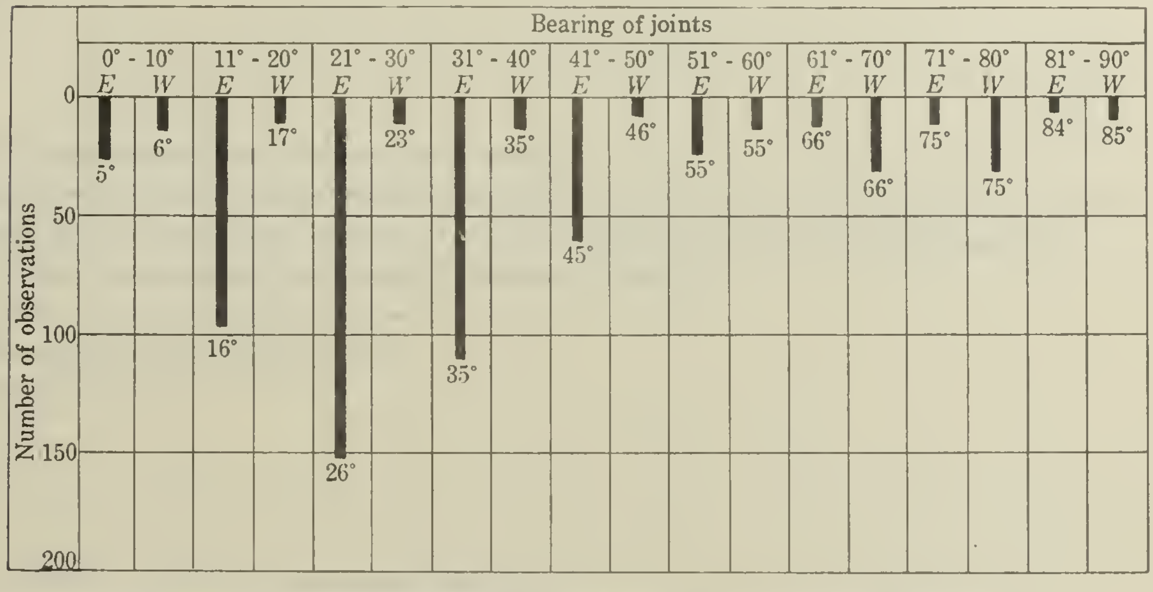


Figure 11.—Unequal distribution of bearings of joints, indicating greater prominence in a few directions

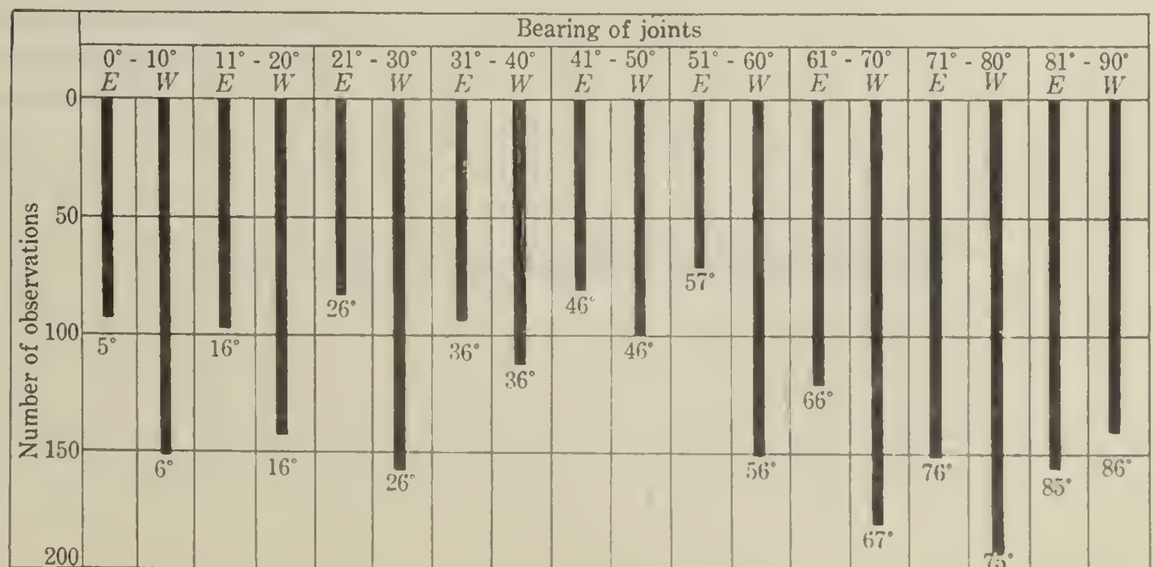


Figure 12.—Combination of Figures 10 and 11

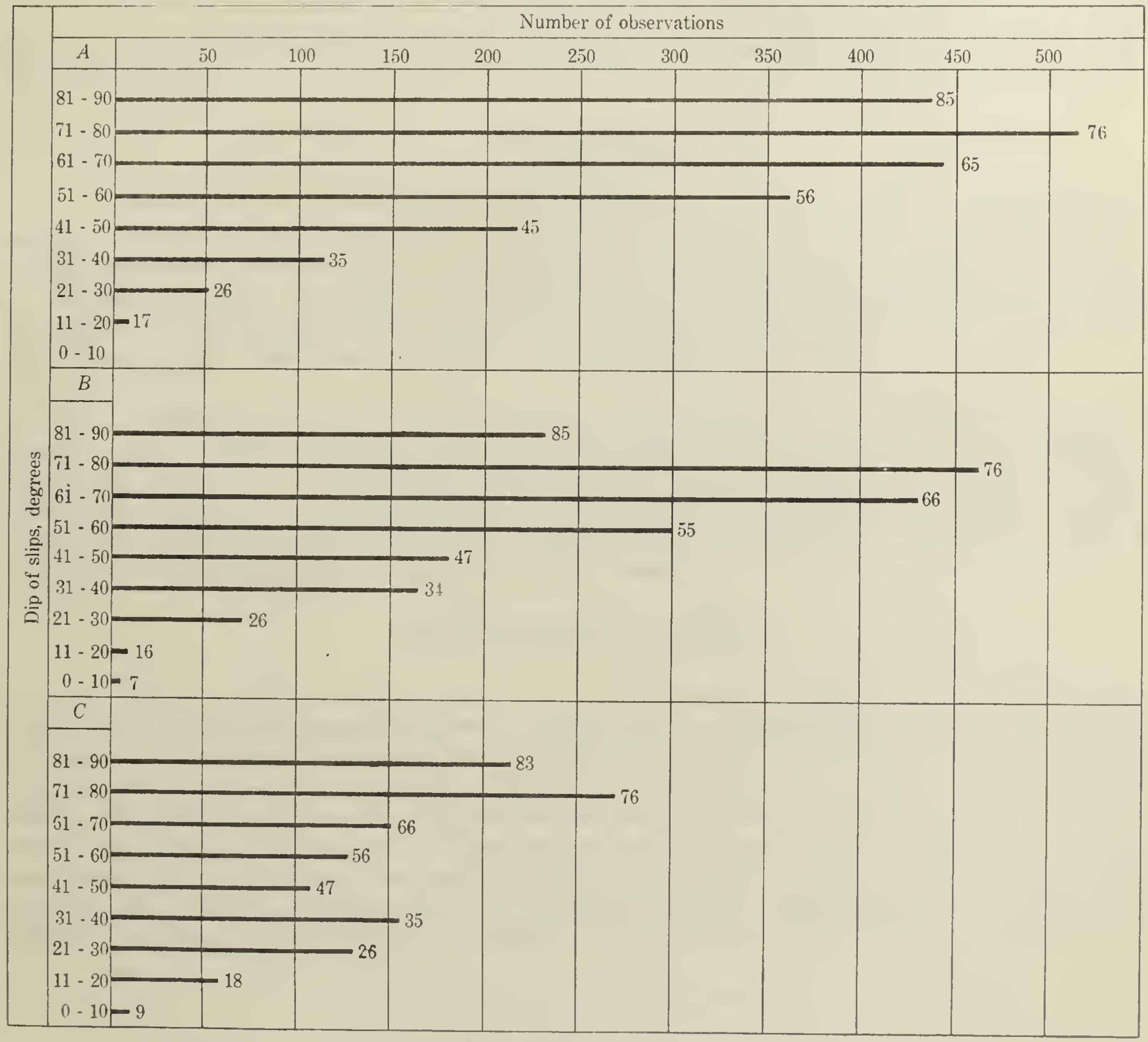


Figure 13.—Diagrams from which angles of draw are determined

The prominence of the bearing of joints in each group is measured by the relative number of readings tabulated in that group. If the prominence is equal in all directions, it may naturally be deduced that jointing is of equal importance in all directions. If, however, certain groups greatly exceed others in prominence, it is concluded that jointing is more effective in those directions.

In the first instance, draw will occur in all directions, whereas in the second instance, it will be more prominent in the direction of the dominating joint planes (see figs. 10, 11 and 12).

Dip of Joints

As the dip of joints is the controlling factor in rock movement, determining the angle of draw, it is quite important that the relation between dip and draw should be thoroughly understood. Observations on dip are taken on the same faces as are those for strike and direction of dip, the data being grouped in columns somewhat similar to the groupings for strike. No attempt is made, however, to introduce the direction of dip, which is considered separately; consequently there is but one column to each group.

Although there is a wide range in types of arrangements of joints according to prominence, yet for the present purpose it is only necessary to mention three as are shown in Figures 13, A, B, and C. In Figure 13B the columns 31 - 40° and 41 - 50° have about equal prominence, whereas in Figure 13C only one column, the 31 - 40°, has an unusual degree of prominence, although the column 21 - 30° is also unusual. The meaning of these variations in prominence is explained under "Interpretation of Results."

Direction of Dip of Joints

The direction of dip of joints is of importance, but need only be considered for those dips that control ground movement; consequently the arrangement of dips according to prominence must be consulted. The most common condition is where the joints have the same angle of dip regardless of direction, a not unusual occurrence with igneous and metamorphic rocks. A similar condition may occasionally be found to exist in bedded deposits that have either been folded or otherwise disturbed by igneous intrusions. With joints dipping at the same angle in all directions the required angle of draw may then be readily applied at any point of the workings, whereas if the direction of movement is more prominent along certain angles of strike, special care must be taken in determining the direction of draw.

The Angle of Draw in Rock Masses

The plane on the angle of draw along which rock masses move or slide is below and back of the broken rock, forming a plane of cleavage between the partly broken and unbroken rock.

The dip of joints determines the angles of draw, but the effective and ultimate draw does not depend entirely upon prominence. The most prominent dips may have high angles, failure by draw taking place upon them after the vertical has been reached by caving. As the angles of draw flatten, the resistance to movement increases until an angle is reached upon which movement of rock masses can not take place (see fig. 6).

The time during which draw may develop lengthens as the angles flatten, unless special and unusual conditions exist that expedite movement. The time element is important therefore, and should be given due consideration in the determination of draw. In determining the ultimate angle of draw, a factor of safety should be taken by selecting an angle somewhat flatter than that determined by observation; higher angles may be chosen if the draw during the life of a mine is alone to be considered.

Interpretation of Data

In the determination of the ultimate angle of draw, the following are controlling factors: (1) prominence of joints, (2) character of joint planes or surfaces, (3) experience in the district, (4) the necessity or desirability of having an adequate factor of safety.

From extended instrument observations in different districts it is obvious that much information has been obtained regarding the physical character of the rock and the condition of the joint planes; in certain localities the planes are clear-cut and smooth, while at other places they are undulating and rough.

In a district where mining has been under way for years with possible subsidence of the surface and the development of draw in the inclosing rocks, considerable information can be obtained relative to the development of draw and its advance from the higher angles to the lowest or ultimate angle of draw. Such information is, however, useful only after all other data has been collected and interpreted; it may reveal unknown and unsuspected weaknesses in the rocks that may take precedence over joints in affecting ground movement.

For safety a slightly smaller angle than the ultimate angle of draw should be chosen, usually from 5 to 10° less according to conditions affecting its choice. Undue caution is not advisable, however, particularly if it means the tying up or loss of considerable ore by employing too low an angle of draw. In some cases the ultimate angle of draw is sufficient safeguard against failure, but with valuable surface property and development openings, an adequate factor of safety should be employed.

The application of data to the determination of actual angles of draw may be shown to best advantage by referring to diagrams embodying actual observations and representing typical occurrences of joints in a variety of rocks. Three cases are cited as representative of conditions in the low-grade copper deposits in porphyry and schists associated with certain barren igneous and metamorphic rocks: (a) joints having dips in regular decreasing order of importance from the high to the low angles, (b) joints having dips with two groups of equal or approximately equal importance, and (c) joints having one group of dips of marked prominence.

The angle of dip of joints is the prime factor in the determination of draw and is, therefore, given special consideration at this point. The strike and direction of dip are merely supplementary to angle of dip and are discussed further on. The application of observation data is discussed in the following cases.

Case A.- Figure 13A shows a normal case - that is one in which the force or forces which produced the joints was regular and well distributed, and further no irregular joints have been formed subsequently. The arrangement shows a systematic decrease from higher to lower dips as represented by the prominence of the respective columns.

Selection from the diagram of the angle of dip that will give the proper angle of draw is more difficult in this arrangement than in most occurrences, as there is a marked regularity in passing from the higher to the lower angles, and it is, therefore, necessary to seek supplementary information. Where mining accompanied by ground movement has been carried on in a locality for many years, ample information is usually available, if properly interpreted. Observations at old abandoned properties and a study of mine maps will reveal the approximate angles at which failure in the rock masses stopped. As a rule, however, such angles are considerably above, that is, higher than the safe angles of draw, due to various irregularities of working and particularly due to the supporting action of waste rock that runs into the workings through cave-ins.

Should the angle obtained from experience in the district indicate an approach to 70°, it is obvious that the angle chosen should not be higher than the average of the next column to the left, namely, 65°. Further, should any group of joints having lower angles be uniformly large, wet, or smoother than the others above and below that group, then it should be chosen, with a factor of 5 or 10° subtracted for additional protection. If the average of such a lower group be 35° then the safe angle of draw would probably be 30°, or if 45°, the draw might well be the average of the next lower group or 35°, thus providing an ample factor of safety.

Assuming that the angle chosen is 35°, that angle can be applied to any direction of working face irrespective of bearing, although the maximum angle of draw will occur where the working face is normal to the direction of dip. The importance of the specially prominent bearings is shown in this connection, and in order to take advantage of them, if any advantage exists, the direction of dip for the 35° column is worked out. If a certain direction of dip predominates then it may be applied to the working faces normal to it, provided there is special reason for additional precautionary measures at such points.

Case B.- A less marked type is where two or more of the groups have prominence considerably greater than the others. The equal or approximately equal prominence of two groups of angles of dip is shown in Figure 13B with the angles 34 and 47° being of approximately equal prominence.

With this arrangement of dips a choice of angle representing the angle of draw can readily be made. Passing the higher angles, 76, 66, and 55°, the angle 47° is reached which would normally be chosen as the angle of draw and an ultimate angle of draw of 40° would be taken as safe. However, as the angle 34° is of nearly equal prominence with that of 47° it is obvious that 34° should be chosen. The application of a factor of safety by reducing the 34° angle by 4° would give ample protection except where the joints in the 34° grouping show excessive weakness through size, character of surface, or wetness.

Case C.- Thrust joints are usually rather low-lying, for instance in the case of "sheeting" in granite. This is well shown in Figure 13C, where the dips of 26 and 35° are of great prominence.

It is obvious that the angle of 35° is the angle that must be chosen, which for safety should be reduced to 30° - the final and ultimate angle of draw. Smoothness or roughness of the joint planes represented in the 35° grouping should be considered, but it is of exceedingly rare occurrence to have rock masses move on an angle lower than 30° - that is, where normal joint planes are involved.

APPLICATION OF DATA ON STRIKE AND DIRECTION OF DIP OF JOINTS

The strikes of joints considered here were taken in igneous and metamorphic rocks and show three types of occurrence: namely (1) those bearing equally in all directions, (2) those bearing principally in a few directions, (3) and those about equally divided between groups one and two (see figs. 10, 11, and 12).

In case (1) as illustrated in Figure 10, the determined angle of draw is applied equally in all directions.

In Case (2), as shown in Figure 11, each prominent group of bearings must receive special consideration. Since movement takes place along the dip of the plane which determines the angle of draw, mine workings parallel with the bearing of the draw plane will have the greatest effect in producing movement.

The combination type which is shown in Figure 12 requires no special discussion. The bearings being prominent in all directions permit the general application of draw to any direction of working face, but should special precautionary measures be desirable, the most prominent bearings should be taken into consideration at points where they are most likely to effect the angle of draw.

The arrangements of the strikes of joints as shown in Figures 10 and 12 and especially that given in the former are particularly favorable to subsidence and ground movement and may be taken advantage of in the application of caving methods in mining.

Summary

Failure of the rock in mine workings is due to insufficient support, and natural support once removed can not be adequately replaced. Failure proceeds upward through the overlying rocks, ultimately breaking through to the surface or more or less seriously affecting it. Under conditions favorable to ground movement, failure extends laterally into the inclosing rock and stops only when resistance to movement exceeds the pressure responsible for failure.

All failure in rocks is contributed to and largely controlled by planes of weakness existing in the rock, the more important being faults, bedding planes, and joints or slips. Which is of the most importance depends upon local conditions of occurrence and prominence. Acting through and in conjunction with these planes of weakness, failure originates in the stopes and breaks to the surface; breaking of rock also extends to the vertical beyond which a sliding action or draw results in further failure until overcome by frictional resistance.

The determination of a definite angle of draw and the direction in which it acts can be readily made by taking observations on the strike, dip, and direction of dip of joints and by analyzing the data obtained.

With definite knowledge of the factors affecting and controlling ground movement the development of properties can be planned for maximum protection of the workings and surface, but the application of such information should not be delayed until failure of the rocks begins.

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

TOPAZ

BY

I. Aitkens

I. C. 6502
September, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

TOPAZ[1]

By I. Aitkens[2]

CONTENTS

INTRODUCTION

It is a common misconception that all yellow stones are topazes and that all topazes are yellow; but neither statement is true. The real topaz is a rather rare mineral, and a large number of yellow stones that masquerade as topaz are nothing more than yellow quartz, known as citrine, but almost universally called topaz. The true topaz is often called "Brazilian" topaz by the jeweler to distinguish it from yellow quartz; nevertheless, both terms are freely applied in trade to other yellow stones.

Topaz is considered the most popular yellow stone for jewelry and is often used as the principal stone in brooches or pendants, especially in old-fashioned ornaments. However, many other species yield pretty yellow stones, some of which are far more numerous and less costly than the topaz.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgement is used: "Reprinted from U. S. Bureau of Mines Information Circular 6502."

2 - Rare metals and nonmetals division, U. S. Bureau of Mines.

DESCRIPTION AND PROPERTIES

Topaz, is a crystalline gem. Chemically it is a fluosilicate of aluminum, in which the fluorine is in part replaced isomorphously by hydroxyl, with the resulting formula, $(Al[F.OH])_2 SiO_4$. Due to this variability in chemical composition the specific gravity varies from 3.4 to 3.6. True or Brazilian topaz has a hardness of 8 on Moh's scale, and is noted for its hardness; very few minerals are harder. It will cut quartz and tourmaline easily, but is very brittle.

Topaz crystallizes in the orthorhombic system, and occurs in prisms with one end regularly terminated. It has a perfect cleavage transverse to the prisms. In contradiction to quartz the prism faces are striated vertically, whereas those of quartz are striated horizontally. Doubly terminated crystals are rare.

When pure the crystals are often perfectly colorless and water-clear, but due to the presence of impurities they may show a wide range of colors--red, yellow, brown, green, or blue. It is an interesting fact that these colors are by no means stable. The brown and wine-yellow fade upon exposure to light, and the sherry-yellow crystals from Brazil assume a fine pink color upon being heated.

USES

Clean, transparent topaz crystals of good color are used as gem stones. Its principal use is for jewelry, but ground topaz is used to a small extent as an abrasive. Powdered topaz is frequently used instead of emery powder for grinding agate, jasper, chalcedony, and other gem minerals.

IDENTIFICATION

Topaz is recognized chiefly by its crystals, its basal cleavage, its hardness, and high specific gravity. It is readily distinguished from citrine because it is harder and heavier. Topaz will scratch citrine and it sinks rapidly in methylene iodide, in which citrine floats. Its perfect cleavage serves to distinguish it from many other precious stones. The cleavage is in only one direction, parallel to the basal plane.

A remarkably clear, colorless transparent crystal might be taken for a diamond, were it not for the fact that it is not nearly so hard and has a much weaker double refractive and dispersing power. In specific gravity, however, topaz ranks close to the diamond.

Topaz is 30 per cent heavier than quartz and is easily distinguished from the common yellow gems beryl, tourmaline, and peridot, which are lighter, and from corundum and garnet, which are heavier.

The minerals commonly called topaz are: the topaz proper; the yellow sapphire known as the "oriental topaz"; and certain quartz minerals known as "Saxon," "Scotch," "Spanish," "smoky," and "false topaz." However, these stones vary widely in hardness and specific gravity as shown by the following table:

	Hardness	Specific gravity
Oriental topaz..........	9	4.01
True topaz...............	8	3.53
Scotch topaz, etc.	7	2.65

SUBSTITUTES

There is very little commercial advantage in manufacturing artificial topaz, since other natural yellow stones are far less expensive and are easier to cut. This accounts for several minerals being commonly called topaz in trade. The principal substitutes to which the name of topaz may be given are yellow quartz, or citrine, and smoky quartz, which are sold as Scotch topaz, false topaz, or smoky topaz, respectively. On the other hand, pink, rose, blue, and greenish topazes, although less common than the yellow variety, are often used in imitation of the more expensive corundum gems. Pink topaz is produced by heating yellow topaz.

HISTORY

The name topaz comes from the Creek word, topazion, meaning to seek. It was applied to gems from an island in the Red Sea, the whereabouts of which were finally located only after some difficulty. Pliny and his contemporaries applied the name to the yellowish peridot found there. In the Middle Ages the term was loosely applied to any yellow stone, but gradually it has been narrowed down to present usage.

MODE OF OCCURRENCE

Since it is usually associated with cassiterite in granitic rocks, topaz is considered a valuable indicator of tin ore. Other minerals commonly associated with it are tourmaline, quartz, fluorite, apatite, beryl, and tungsten ores.

According to Dana, topaz occurs in cavities or rhyolite lavas and granite; in pegmatite veins, especially those carrying tin; and at times is also found as rolled pebbles in stream sands (placer deposits). It is formed through the agency of fluorine-bearing vapors given off during the last stages of the solidification of igneous rocks.

MINING METHODS

In Brazil the principal mine- the Boa Vista- had been worked for many years by open-cut methods. A pit was first dug for some distance down into the overburden formed by slides and the caving of the decomposed inclosing

rock, and from the bottom of this excavation small inclined shafts were sunk to reach the topaz-bearing deposit. By this method the gem-bearing clay could be followed down some meters below the drainage level, but the mine could not be worked successfully except during the dry season.

In 1908 when these old topaz mines were reopened, deep mining methods were employed at this property and have been introduced also into other localities in Brazil.

The presence of water within a few feet of the surface has likewise hindered the working of prospects in the United States. In this country the pits were ordinarily from 8 to 15 feet deep and about 25 feet wide.

DOMESTIC PRODUCTION

The production of topaz in the United States has never been large. Although the mineral is widely distributed through the States, crystals of gem quality are somewhat rare. The majority of domestic stones are colorless, but some fine blue and bluish-green crystals have been found. Production statistics for the United States are not available, however, after 1921, when the United States Geological Survey discontinued the canvass of producers.

Table 1.- Value of topaz produced in the United States, 1906-1921[1]

Year	Value	Year	Value	Year	Value
1906	$1,550	1912	$ 375	1918	$ 907
1907	2,300	1913	736	1919	210
1908	4,435	1914	1,380	1920	767
1909	512	1915	862	1921	2/
1910	884	1916	1,005		
1911	2,675	1917	230		

1/ Compiled from Mineral Resources of the United States. Part II. (annual chapters). 2/ Less than three producers; figures not published separately

DOMESTIC DEPOSITS

Crystals from Harndon Hill, in the vicinity of Stoneham, Maine, were identified as topazes in 1882 and among the more important subsequent finds may be mentioned: on Baldface Mountain, near North Chatham, N. H., in 1888; in San Diego County, Calif., about 1903; in Mason County, Tex., in 1904. None of these deposits nor others in Maine, Colorado, and Utah have ever been worked except intermittently. Topaz has been found also in Georgia, Idaho, Montana, and Wyoming.

California

In California, topaz deposits were discovered 4 miles from Ramona in San Diego County. This locality produced rich gems as well as forms suitable for beautiful cabinet specimens. These came principally from the Little Three mine.

The Mountain Lily Gem Mine on Smith or Aquanga Mountain in San Diego County also produced some large topaz crystals. One of these, a decidedly green stone, weighed $3\frac{1}{4}$ pounds and exhibited a number of crystal faces. This topaz was found with other greenish and white topaz crystals in a pegmatite ledge. Tourmaline was found in pockets in the same ledge but not in the same pockets as the topaz. A part of the $3\frac{1}{4}$-pound crystal was transparent and of good gem material. Some time later, 30 crystals of bluish topaz were taken from the Mountain Lily mine, near Oak Grove in San Diego County.

Georgia

About 1913, two gems were cut from crystals found in the Williams mica mine near Two Run, Ga. One of these turned out to be ordinary quartz, but the other proved to be colorless topaz. This was found inclosed in a cavity in a large crystal of mica, a new locality for topaz and an unusual mode of occurrence.

Idaho

White topaz was discovered about 1919, at City of Rocks, 5 miles northwest of Moulton, Cassia County, Idaho. This mineral was cut into stones of 1 to 3 carats and is said to resemble the diamond. Stones exhibited by the owner of these claims were exceptionally clear and could cut glass like a high-priced diamond would.

Maine

According to Kunz[3] most of the topaz crystals from Harndon Hill, in the southwest corner of the town of Stoneham, Me., were found in one pocket with clevelandite. These stones ranged in size from quite small crystals to large, rough, opaque masses, weighing 10 to 20 kilograms. Some were transparent only in parts, while the better crystals were colorless or faintly tinted with green or blue and measured 10 to 60 millimeters across. This locality has been described by E. S. Bastin.[4]

Montana

In 1920 topaz was found about 18 miles southeast of Butte, Mont. Sufficient work was done on these claims to prove that it was good gem material.

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3 - Kunz, Geo. F., Topaz and Associated Minerals at Stoneham, Me.: Am. Jour. Sci., 3d ser., vol. 27, 1884, pp. 212-216.

4 - Bastin, E. S., Geology of the Pegmatites and Associated Rocks of Maine: U. S. Geol. Survey Bull. 445, 1911, pp. 100-102.

New Hampshire

According to Kunz,[5] in May, 1888, E. A. Andrews discovered topaz on Baldface Mt., about 4 miles northwest of North Chatham, N. H. No systematic mining for topaz and associated minerals was done for some time, although occasional crystals were picked up by prospectors and mineral collectors working only a few days at a time. Later, however, considerable systematic prospecting was done on Baldface Mt. by John Chandler of North Chatham.

In this locality the topaz occurs in crystals up to more than an inch thick. Some perfectly transparent crystals were found, but the majority of them were merely translucent or were transparent only in places. Although some of these stones were suitable for gems, most of them were worth more as specimens because of the perfection of the crystals. They ranged in color from colorless to pale bluish-green, but some were yellow from iron stains on the surface.

Texas

In 1904 an important deposit of gem topaz was accidentally discovered in Mason County, Tex., by R. L. Parker of Streeter. Since then, topaz has been found near Streeter and also near Katemcy, in Mason County, the former is 8 miles due west and the latter 12 miles north of Mason, the county seat.

Meyer[6] has described the deposits of topaz in Mason County, Tex. An interesting discovery made by Meyer was that stream tin or cassiterite was associated with the topaz in the concentrates from the dry placers. The tin mineral was found in broken crystals and angular grains of resin-yellow to brownish-black color.

According to Meyer, the topaz occurs in pegmatite, but with different associations at the two localities. Near Streeter the crystals are found associated with microcline, feldspar, biotite, tourmaline, smoky quartz, and albite. Some 200 pounds of good topaz crystals had been obtained, and besides the clear, colorless crystals a small number of beautiful light-blue ones had been found. Meyer describes the largest crystal found here as about 3 inches in diameter with a faint greenish-blue color.

At the deposit near Katemcy, the topaz is intimately associated with quartz and feldspar, and the interstices are filled with a reddish felsitic rock. These crystals range up to an inch or more in thickness and are frozen in the rock. Meyer further states that about 80 per cent of the

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5 - Kunz, Geo. F., Gems and Precious Stones of North America: New York, 1890, p. 70.

6 - Meyer, H. Conrad, Topaz and Stream Tin in Mason County, Texas: Eng. and Min. Jour., Mar. 8, 1913, pp. 511-512.

mass of this rock is topaz, but that a much smaller proportion of the topaz from this place is suitable for gem purposes than at the other locality.

Utah

Although in 1859, a geologist, Henry Engelmann, first discovered the occurrence of topaz in the Thomas Range, Utah, little was heard of this region until 1884. Numerous specimens were collected at this time by Prof. J. E. Clayton of Salt Lake City, and collectors have since visited the locality and brief descriptions have been written. The following notes have been abstracted from an article by Horace B. Patton[7] on the occurrences in the Thomas Range.

These deposits are 40 miles north of Sevier Lake and a little over 40 miles northwest of Deseret. This locality is known as Topaz Mountain and is that portion along the southeast side where topaz crystals have been found most abundantly. The rocks of this portion of the Thomas Range are of volcanic origin and the only sedimentary rock exposed near this location is a bluish-gray limestone.

The three types of crystals from this locality are (1) fine transparent crystals which occur principally in lithophysae cavities; (2) rough opaque crystals which occur scattered through the solid rhyolite; and (3) smooth opaque crystals which are similar to the rough opaque, except that the faces are smooth and better developed.

These topaz crystals are present over a large area, but abundant over a limited area only. Weathering of the topaz-bearing rhyolite has left many crystals scattered over the surface; most of them are very small but brilliant. A few large transparent crystals are found, but the tiny ones shine so brilliantly in the sunlight as to make it difficult to locate the larger crystals.

According to Alling[8] the transparent crystals make very brilliant gems, when found of sufficient size for cutting. These are perfectly colorless and are sold mostly as souvenirs to tourists under the name of "white topaz." They are also highly prized for collection specimens on account of their transparency and the quality of the crystal faces.

Wyoming

Crystals of topaz were obtained from the headwaters of Bighorn River in northern Wyoming. These displayed a development of a number of faces including prisms, pyramids, domes, pinacoids, and base. Though mostly small, all of these crystals were transparent and colorless, resembling in quality

7 - Patton, Horace B., Topaz-bearing Rhyolite of the Thomas Range, Utah: Bull. Geol. Soc. Am., vol. 19, 1908, pp. 177-192.

8 - Alling, A. N., Topaz from the Thomas Range, Utah: Am. Jour. Sci., 3d ser., vol. 33, 1887, p. 146.

the colorless topaz from the Thomas Range, Utah. These are of interest as specimens because of their quality and sharp crystal form, but would not have much value as gems except for local souvenir trade.

THE INDUSTRY IN FOREIGN COUNTRIES

Brazil and Ceylon are the two principal sources of topaz to-day. However, topaz of fine gem quality has been produced in small quantities in many other localities. Fine blue and sherry-colored crystals have been found in Siberia; blue ones in Scotland and Ireland; blue and white ones in Ceylon; good crystals of pale blue and green colors have come from the Ural Mountains and from Nerchinsk in Siberia; and some of a rare reddish hue have come from Russia. A large, rough crystal weighing 137 pounds of opaque topaz was obtained from a feldspar quarry in Norway. Australia, Japan, and Mexico have also produced topaz of gem quality.

Brazil

The principal localities in Brazil are, Ouro Preto, Villa Rica, and Minas Novas in the State of Minas Geraes. At all of these places stones of lovely shades of blue and wine-yellow occur. Reddish stones also have been found in Ouro Preto. The sherry-yellow crystals used so extensively in jewelry and for which Brazil has become noted, have come mostly from the neighborhood of Ouro Preto, and is that material which supplied the pink ("burnt") topaz. Orville Derby[9] has described the occurrences near Ouro Preto, Brazil.

According to Kunz,[10] one of the largest single pieces of topaz in the world has been added to the Smithsonian Institution collection from Brazil. This piece weighs 95 pounds. At least six sections of transparent topaz of a very pale yellow or white have come to the United States from Brazil; three of these pieces apparently fitted together and formed a crystal of at least 300 pounds.

Ceylon

In Ceylon, topaz is fairly abundant, but the bright yellow variety is almost entirely absent. The true topazes are either colorless or light green, but occasionally pinkish-yellow or yellow-brown stones are also found. The colorless variety is frequently cut and sold under the name of water-sapphire, but the name is entirely misapplied, as the true water-sapphire is blue; green stones are sold with true beryls as aquamarines; the yellowish-pink stones closely resemble the pink stones of Brazil, but with the curious difference that while the Brazilian stones turn to a clear pink on being heated, the Ceylon stones are absolutely decolorized by heat.

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9 - Derby, Orville A., Topaz near Ouro Preto, Brazil: Am. Jour. Sci., 4th ser., vol. 11, Jan., 1901, pp. 25-34.

10 - Kunz, Geo. F., Precious and Semi-precious Stones: Min. Ind., 1926, p. 586.

Burma

Some unusually fine specimens have come from the great pegmatite dike at Sakangei, Burma. According to Kunz,[11] one particular crystal of a rich cinnamon-brown, beautifully crystallized, came from this locality. This crystal weighed 1088.5 grams (35 troy oz.), and measured 95 millimeters in width, 90 millimeters in depth, and 82 millimeters in height. Doubly terminated crystals have also been found in this locality. Adams and Graham[12] have described the topaz occurrences at Sakangei, Burma.

Australia

Topazes of gem quality have been found in Western Australia, Tasmania, and New South Wales. About 1902, E. L. Simpson,[13] mineralogist of the Geological Survey of Western Australia, announced the discovery of topaz in Western Australia. This deposit was found in the Coolgardie gold field near the town of Londonderry and in a dike of very coarse pegmatite, traversing amphibolite rock. The stones procured here were so light in color as to appear like white topazes.

In Tasmania the well-known localities for topaz are Mount Cameron, Flinders Island, and Bell Mount. Tasmania is noted for its remarkably fine water-clear topaz. These stones are cut and polished for ornamental purposes. Topaz also occurs here in shades of sea-green and blue. Some of the crystals from Flinders Island are of remarkable size, up to several inches in diameter.

The principal deposits of New South Wales are to be found at Emmaville and Oban. These deposits are described and detailed accounts are given in the Records of the Australia Museum by Anderson.[14]

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I. C. 6503 June, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

MINING METHODS AND COSTS AT METAL MINES OF THE UNITED STATES

BY

CHARLES WILL WRIGHT

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT METAL MINES OF THE UNITED STATES[1]

By Charles Will Wright[2]

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2 Chief engineer, mining division, U. S. Bureau of Mines.

INTRODUCTION

The purpose of this paper is to present briefly certain preliminary results of the study of mining methods by the United States Bureau of Mines; to show the relative importance of each method in terms of tonnages mined and man-hours consumed; to give to the mining engineers and particularly to the young student of mining a better understanding of the principal mining methods; and to indicate changes now in progress and to present suggestions for improvements in the art of mining.

Questions of mine development and details of underground practice such as drilling and blasting, loading, hauling, hoisting, ventilating, pay systems and mine management are omitted, as these have been or will be discussed in other bureau publications.

Statistics show that only about 400 metal mines in the United States produced $100,000 or more in 1929. These mines represented about 90 per cent of the total metal production. The study is therefore not an appalling one, as some may think, but a relatively limited one.

ACKNOWLEDGMENTS

The writer is indebted to C. F. Jackson and J. B. Knaebel for their assistance and to the economics and health and safety branches of the United States Bureau of Mines for their help in securing the statistical data. A special expression of thanks is due to the mine officials who gave so generously of their time to prepare detailed reports on their mining operations, which reports, published by the bureau as Information Circulars, are the basis of this study. A list of these Information Circulars and their authors is added at the end of this paper, each paper having a reference number, as (1), etc., used throughout the text and in most of the tables.

ORE OCCURRENCE

The engineer is interested not only in the size, shape, dip, and metal content of an ore deposit to be mined, but also in the distribution of the metal-bearing minerals in the deposit. These minerals sometimes occur in rich shoots or concentrated in small masses or areas, and in such cases selective mining and hand sorting usually give the best results. There are instances of failures resulting from attempts to mine and mill an entire orebody, where smaller-scale operations, requiring less plant and equipment and using selective mining and hand sorting, might have succeeded.

The nature of the mineral, whether a sulphide or oxidized ore or a combination of the two, has an important bearing on its ultimate value. The fineness and associations of the mineral particles in the ore must also be determined, and neglect of this may result in a failure. Certain copper-zinc and lead-zinc ores are so finely intermixed that their separation, even by the finest grinding, is only partial, and only at a high cost is it possible to make a commercial product from them.

Next to the characteristics of the deposit, the conditions of the inclosing rocks should be investigated. The strength of the roof or walls as well as that of the orebody itself must be considered in determining the ultimate value of an ore and the mining method

that should be used. Some ore deposits have no sharply defined walls as the ore minerals are disseminated in the country rock, and the limits of minable ore are determined by the amounts of ore minerals present. These limits vary with metal prices and improvements in the technique of mining and milling. It is from such deposits that to-day some of our largest production is coming.

CLASSIFICATION OF MINING METHODS

Classifications of stoping methods have been published from time to time in the technical press and in the bulletins of various professional associations. These classifications have been based upon the method of support, direction of working, sequence of operations, methods of handling broken ore, and combinations of these methods. Charts have been prepared, including all variations and combinations of methods, showing that upwards of one hundred and fifty such variations and combinations are in use. These variations have resulted in confusing somewhat the fundamental distinctions between the principal methods.

Certain methods extensively employed in the Southwest, and there considered to be distinct, may be thought by the engineer in the Lake Superior region to be of secondary importance and not to be included in a simplified classification. It would seem, therefore, that the simpler and more condensed the classification, the more readily and generally it will be accepted.

In the last analysis, the method to be applied is governed primarily by those characteristics of the deposit which determine the area of back and size of excavation which will be self-supporting during the period of stoping, and by the nature, size, and spacing of supports required to maintain these openings, and by the requirements as to permanently supporting the surface against subsidence. This is true whether the deposit be large or small, flat, inclined or vertical, and whatever may be the grade of ore and distribution of minerals. These last factors are recognized as having an important bearing upon the details and variations of the principal method selected, but primarily the method to be employed hinges upon the question of support. It is therefore logical to base a classification of mining methods upon the method of support, as follows:

Stopes naturally supported:

1. Open-stope method, including open stopes with casual pillars or with regular arrangement as in the room-and-pillar method and the sublevel-stoping method.

Stopes artificially supported:

2. Shrinkage method, with or without pillars.
3. Cut-and-fill method, with or without temporary supports.
4. Square-set or stull-set method.

Caved stopes:

5. Caved-stope methods, including top slicing, sublevel caving and block caving.

Combined underground methods:

6. Combinations of supported and caved-stoping methods.

Surface mining:

7. Opencut methods.

The above classification does not take into consideration the direction of stoping as "underhand" or "overhand," "horizontal or rill," the sequence of working as "advancing" or "retreating," or the methods of handling as by "chutes," "branch raises," "scrapers," or "hand shoveling," although these are recognized as important variations of the principal methods.

In this classification "stopes naturally supported" includes all stopes where the excavated ore is not replaced by other support _during the period covered by mining operations in the stope_. Props and stulls may be used in such stopes, however.

"Stopes artificially supported" includes all those stopes in which the excavated ore usually is replaced by filling (ore or waste) or by square-set or stull-set timbering. Where supported by broken ore during stoping operations, the stopes are not always filled after the ore has been removed.

The entire classification includes six principal methods, and to these has been added a seventh, namely "combined underground methods," to include stoping operations where two methods are systematically employed together in a regular scheme for mining each block of ore, the principal method being dependent for its success upon the application of the other or subordinate method.

Again, at some mines a portion of the ore deposit is mined by the square-set method and another by cut-and-fill, or the upper portion by open-pit methods and the lower portion by shrinkage; at some mines different methods are used for different portions of the same orebody, and in classifying these mines the method is chosen by which the greatest tonnage is produced. Two independent methods used on different portions of the same deposit are not considered combined methods.

In the above classification all variations of methods not widely and generally understood are deliberately omitted, and it is believed that all known methods of stoping will fall logically under one of the seven captions of this classification.

RELATIVE IMPORTANCE OF EACH MINING METHOD AS APPLIED TO METAL MINES IN THE UNITED STATES

Two tables have been prepared; the first shows the tonnages of each metal-bearing ore mined by each method, and the percentages of the total tonnage; the second shows the man-hours worked in the mines according to the metal produced, accident data according to the mining method used, and the percentage of labor employed for each method. A comparison of these two tables brings out the following points of interest.

Table 1 shows that in 1929 the total output of these $100,000 mines was 169,969,366 tons from 405 mines. Forty-two per cent of this total was mined from opencuts, 21 per cent was produced by the open-stope method, and the caved-stope methods were responsible for 25 per cent.

Table 2 shows that the total man-hours worked, 162,645,375, is about the same as the total tonnage mined, indicating that approximately 1 ton was mined for each man-hour worked. The square-set method consumed 25 per cent of the total labor, to produce 4.6 per cent of the total tonnage, while the open-stope method produced 21 per cent of the tonnage with the same labor consumption as in the square-set method. The opencut method was responsible for 42 per cent of the tonnage, with a consumption of only 14 per cent of the total labor. Fifteen times more tonnage was mined per man-hour by the open-pit method than by the square-set method.

The accident records show that the opencut, sublevel-caving and top-slicing, and open-stope methods had the lowest accident rates or best records, and the square-set, cut-and-fill, block-caving and shrinkage methods had the highest accident rates or worst records.

Table 1.- Tonnages classified according to mining method at metal mines with individual production of more than $100,000 in 1929

Mining method	No. mines	Gold ore, tons	No. mines	Copper ore, tons	No. mines	Silver, lead, and zinc ore, tons	No. mines	Iron ore, tons	No. mines	Total tons, ore	Per cent
Square-set	7	379,022	26	4,318,294	28	3,121,746	–	–	61	7,819,062	4.6
Cut-and-fill	2	13,150	6	1,573,135	11	664,252	2	305,575	21	2,556,112	1.5
Shrinkage	11	5,645,110	12	2,306,930	10	2,011,500	4	1,070,608	37	11,034,148	6.5
Open-stope	11	364 516	12	6,121,545	113	15,502,250	34	13,298,362	170	35,186,673	20.8
Top slicing	–		–	–	–		37	15,904,288	37	15,904,288	9.4
Sublevel caving	–	–	–	–	–		17	7,303,976	17	7,303,976	4.3
Block caving	–	–	7	18,576,196	–	–	–	–	7	18,576,196	10.9
Open-pit	–	–	7	31,605,847	–	–	48	39,883,064	55	71,488,911	42.0
Total	31	6,401,798	70	64,501,947	162	21,299,748	142	77,765,873	405	169,969,366	100.0

1 Reduced to short-ton basis.

Table 2.- Man-hours worked and accident data classified according to mining method at metal mines with individual production of more than $100,000 in 1929

Mining method	Man-hours worked						Accidents per thousand 300-day workers	
	Gold	Copper	Silver, lead, and zinc	Iron	Total	Per cent	No. killed	No. injured
Square-set	2,028,328	21,312,106	17,004,829	–	40,345,263	24.8	3.91	390
Cut-and-fill	207,600	6,628,064	3,642,496	612,712	11,090,872	6.8	4.09	223
Shrinkage	3,916,770	5,584,400	4,175,792	997,000	14,673,976	9.0	5.37	302
Open-stope	2,018,640	8,370,798	18,011,090	15,232,038	39,965,832	24.6	3.26	258
Top slicing	–	–	–	14,826,200	14,826,200	9.1	3.15	74
Sublevel caving	–	–	–	6,922,448	6,922,448	4.3	2.39	57
Block caving	–	11,679,624	–	–	11,679,624	7.2	4.81	219
Open-pit	–	9,097,408	–	14,045,752	23,143,160	14.2	2.18	112
Total	8,171,338	62,672,400	42,834,207	52,636,150	162,647,375	100.0	3.50	238

AVERAGE RESULTS OF EACH MINING METHOD, CLASSIFIED ACCORDING TO METALLIC MINERAL MINED

A series of tables has been prepared to show the tonnages and metal content of ores mined by each method, the man-hours worked, and the tonnage of ore and units of metal produced per man-hour at the gold, copper, lead-zinc, and iron mines. In reviewing these tables, one will note the amounts of ore of each metal mined by each method; and, having the average output per man-hour, one can easily estimate the total amount of labor employed at the mines if the ore or metal production is known, or vice versa.

GOLD MINES -- 1929

All mines in the United States with individual output valued at over $100,000

Mining method	No. of mines	Man-hours	Tons mined	Per cent of total tonnage	Yield, ounces per ton	Man-hours: Per ton	Man-hours: Per ounce output
Square-set	7	2,028,328	379,022	5.9	0.35	5.35	15.3
Cut-and-fill	2	207,600	13,150	.2	1.03	15.78	15.3
Open-stope	11	2,018,640	364,516	5.7	.42	5.54	13.2
Shrinkage	11	3,916,770	5,645,110	88.2	.17	.69	4.01
Total	31	8,171,338	6,401,798	100.0	0.20	1.27	6.41

COPPER MINES -- 1929

All mines in the United States with individual output valued at over $100,000

Mining method	No. of mines	Man-hours	Tons mined	Per cent	Yield, pounds per ton	Mam-hours: Per ton mined	Mam-hours: Per pound output
Square-set	26	21,312,106	4,318,294	6.7	93.0	4.95	0.054
Cut-and-fill	6	6,628,064	1,573,135	2.4	67.5	4.18	.062
Shrinkage	12	5,584,400	2,306,930	3.6	33.6	2.45	.073
Open-stope	12	8,370,798	6,121,545	9.5	26.7	1.37	.051
Block caving	7	11,679,624	18,576,196	28.8	18.6	.63	.034
Opencut	7	9,097,408	31,605,847	49.0	20.6	.29	.014
Total	70	62,672,400	64,501,947	100.0	27.4	0.95	0.034

LEAD-ZINC MINES -- 1929

All mines in the United States with individual output valued at over $100,000

Mining method	No. of mines	Man-hours	Tons mined	Per cent	Yield, pounds per ton	Man-hours: Per ton mined	Man-hours: Per pound output
Square-set	28	17,004,829	3,121,746	14.7	246.6	5.45	0.022
Cut-and-fill	11	3,642,496	664,252	3.1	161.7	5.48	.034
Shrinkage	10	4,175,792	2,011,500	9.4	144.4	2.04	.015
Open-stope	113	18,011,090	15,502,250	72.8	68.2	1.12	.017
Total	161	42,834,207	21,299,748	100.0	91.3	2.01	0.022

IRON MINES -- 1929

All mines in the United States with individual output valued at over $100,000

Mining method	No. of mines	Man-hours underground	Tons mined[1]	Per cent	Production, of iron tons	Yield per ton mined, tons	Man-hours Per ton mined	Man-hours Per ton ron
Cut-and-fill	2	612,712	272,835	0.4	149,369	0.550	2.25	4.09
Shrinkage	4	997,000	955,900	1.4	435,145	.456	1.04	2.28
Open-stope	34	15,232,038	11,873,538	17.1	4,933,589	.415	1.28	3.08
Top slicing	37	14,826,200	14,200,257	20.5	7,399,874	.520	1.05	2.02
Sublevel caving	17	6,922,448	6,521,407	9.4	3,444,545	.529	1.06	2.00
Opencut	48	14,045,752	35,609,878	51.2	16,877,752	.475	.39	.82
Total	142	52,636,150	69,433,815	100.0	33,240,274	.478	0.76	1.60

1 Mines represented = 33,140,274 long tons iron
Total output, 1929 = 37,601,850 long tons iron.

CONSUMPTION OF LABOR, POWER, AND EXPLOSIVES, AND AVERAGE COSTS PER TON MINED BY EACH METHOD

A review of the following table brings out rather strikingly the advantages or disadvantages of the different methods of mining with reference to consumption of labor, power, and explosives, as well as to mining costs. These consumption figures and costs show wide variations, in some instances as much as a thousand per cent.

The table shows the average of the mines from which we have specific data, and the number of mines represented is indicated in each case. Of special interest is the comparison of man-hours per ton with mining costs. One might almost make the formula:

Cost of mining = man-hours per ton x $1.

As costs vary with the prices paid for labor and supplies and with locality, the dollars and cents figures are not as important for comparative purposes as the expenditure in units of labor, power, explosives and other supplies per ton mined or per unit of metal produced.

One might ask, "Why give costs in dollars, especially average costs?" No mine can be considered an average mine. Costs often vary widely at the same mine from year to year due to greater or less development work, change in rate of production, depth of workings, amount of water to be handled, etc. Again, a direct comparison of costs of two mines using the same method is misleading, as no two mines have the same size and shape of orebody, grade of ore, strength of wall rocks, or other operating conditions. However, the average costs do indicate in a general way the difference in cost of mining by the different methods; these are given below for a limited number of mines.

Average results for different mining methods

Mining method	Man-hours per ton, all underground labor	Tons per man-shift, all underground labor	Explosive used per ton, pounds	Power, kw.h. per ton	Total underground cost per ton
Square-set	5.15	1.553	1.363	34.9	$4.825
Average of	61 mines	61 mines	10 mines	19 mines	11 mines
Cut-and-fill	4.34	1.843	.660	10.045	$3.076
Average of	21 mines	21 mines	5 mines	4 mines	5 mines
Shrinkage	1.33	6.015	1.656	15.33	$2.682
Average of	37 mines	37 mines	18 mines	16 mines	16 mines
Open-stope	1.17	6.837	.844	8.90	$1.250
Average of	170 mines	170 mines	22 mines	17 mines	19 mines
Sublevel caving	.95	8.421	.605	13.00	$1.438
Average of	17 mines	17 mines	4 mines	2 mines	2 mines
Top slicing	.932	8.584	.520	7.252	$1.208
Average of	37 mines	37 mines	13 mines	12 mines	3 mines
Caving	.628	12.739	.244	3.0	.516
Average of	7 mines	7 mines	3 mines	3 mines	4 mines
		Surface			
Opencut	.324	24.691	.308	2.71	.309
Average of	55 mines	55 mines	3 mines	3 mines	3 mines

The following table shows in a rough way the relative consumption of explosives and power according to metal mined; the number of mines from which the data were obtained is indicated in each case. With data from more mines, these averages would be more representative.

Explosives and power consumption according to metal mined

Metal	Tons of ore represented	Explosives, pounds per ton	Power, kw.h. per ton
Iron:			
30 underground mines	14,298,700	0.548	8.38
Lead and zinc:			
15 underground mines	3,159,701	.920	17.65
Copper:			
10 underground mines	11,293,237	.439	7.64
3 open-pit mines..	21,246,804	.308	2.71
Gold:			
8 underground mines	5,554,410	.844	9.72

IRON MINES -- 1929

All mines in the United States with individual output valued at over $100,000

Mining method	No. of mines	Man-hours underground	Tons mined[1]	Per cent	Production, of iron tons	Yield per ton mined, tons	Man-hours Per ton mined	Man-hours Per ton ron
Cut-and-fill	2	612,712	272.835	0.4	149,369	0.550	2.25	4.09
Shrinkage	4	997,000	955,900	1.4	435,145	.456	1.04	2.28
Open-stope	34	15,232,038	11,873,538	17.1	4,933,589	.415	1.28	3.08
Top slicing	37	14,826,200	14,200,257	20.5	7,399,874	.520	1.05	2.02
Sublevel caving	17	6,922,448	6,521,407	9.4	3,444,545	.529	1.06	2.00
Opencut	48	14,045,752	35,609,878	51.2	16,877,752	.475	.39	.82
Total	142	52,636,150	69,433,815	100.0	33,240,274	.478	0.76	1.60

1 Mines represented = 33,140,274 long tons iron
Total output, 1929 = 37,601,850 long tons iron.

CONSUMPTION OF LABOR, POWER, AND EXPLOSIVES, AND AVERAGE COSTS PER TON MINED BY EACH METHOD

A review of the following table brings out rather strikingly the advantages or disadvantages of the different methods of mining with reference to consumption of labor, power, and explosives, as well as to mining costs. These consumption figures and costs show wide variations, in some instances as much as a thousand per cent.

The table shows the average of the mines from which we have specific data, and the number of mines represented is indicated in each case. Of special interest is the comparison of man-hours per ton with mining costs. One might almost make the formula:

Cost of mining = man-hours per ton x $1.

As costs vary with the prices paid for labor and supplies and with locality, the dollars and cents figures are not as important for comparative purposes as the expenditure in units of labor, power, explosives and other supplies per ton mined or per unit of metal produced.

One might ask, "Why give costs in dollars, especially average costs?" No mine can be considered an average mine. Costs often vary widely at the same mine from year to year due to greater or less development work, change in rate of production, depth of workings, amount of water to be handled, etc. Again, a direct comparison of costs of two mines using the same method is misleading, as no two mines have the same size and shape of orebody, grade of ore, strength of wall rocks, or other operating conditions. However, the average costs do indicate in a general way the difference in cost of mining by the different methods; these are given below for a limited number of mines.

Average results for different mining methods

Mining method	Man-hours per ton, all underground labor	Tons per man-shift, all underground labor	Explosive used per ton, pounds	Power, kw.h. per ton	Total underground cost per ton
Square-set	5.15	1.553	1.363	34.9	$4.825
Average of	61 mines	61 mines	10 mines	19 mines	11 mines
Cut-and-fill	4.34	1.843	.660	10.045	$3.076
Average of	21 mines	21 mines	5 mines	4 mines	5 mines
Shrinkage	1.33	6.015	1.656	15.33	$2.682
Average of	37 mines	37 mines	18 mines	16 mines	16 mines
Open-stope	1.17	6.837	.844	8.90	$1.250
Average of	170 mines	170 mines	22 mines	17 mines	19 mines
Sublevel caving	.95	8.421	.605	13.00	$1.438
Average of	17 mines	17 mines	4 mines	2 mines	2 mines
Top slicing	.932	8.584	.520	7.252	$1.208
Average of	37 mines	37 mines	13 mines	12 mines	3 mines
Caving	.628	12.739	.244	3.0	.516
Average of	7 mines	7 mines	3 mines	3 mines	4 mines
		Surface			
Opencut	.324	24.691	.308	2.71	.309
Average of	55 mines	55 mines	3 mines	3 mines	3 mines

The following table shows in a rough way the relative consumption of explosives and power according to metal mined; the number of mines from which the data were obtained is indicated in each case. With data from more mines, these averages would be more representative.

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Metal	Tons of ore represented	Explosives, pounds per ton	Power, kw.h. per ton
Iron:			
30 underground mines	14,298,700	0.548	8.38
Lead and zinc:			
15 underground mines	3,159,701	.920	17.65
Copper:			
10 underground mines	11,293,237	.439	7.64
3 open-pit mines..	21,246,804	.308	2.71
Gold:			
8 underground mines	5,554,410	.844	9.72

FACTORS TO BE CONSIDERED WHEN SELECTING A MINING METHOD FOR A GIVEN ORE DEPOSIT

In considering the possible commercial exploitation of any ore deposit, its probable tonnage, grade, and production-costs must be taken into account. The mining method employed in the extraction of the ore will determine largely the production-costs. The method must naturally be the safest, most efficient and therefore the most economical, since an improper selection may result in high costs, low extraction, and loss of life.

The financial position of the mining company may be a factor in determining not only the amount of preliminary development work, but also the mining method to be used. Adequate development, particularly in depth and extent to the limits of the ore deposit or property line, although an initial drain on capital, usually results in greater flexibility and safety in mining, as well as in economy and longer life.

The stoping method best adapted to the extraction of an orebody depends upon a number of factors, such as the physical characteristics of the orebody and of the inclosing rocks, and the grade of ore and distribution of valuable minerals therein. These physical factors, as previously referred to, are the size, shape, dip, and strength of the orebody and the strength of the roof or walls of inclosing rock. The distribution of the ore minerals in the orebody will determine whether wholesale or selective mining should be applied. Low-grade deposits, where large-scale methods of extraction may be used, are often more profitable to mine than small high-grade deposits, where more costly methods are necessary.

The availability and cost of timber and material for use as waste-fill may have an important bearing on the choice of a stoping method! also the chemical nature of the ore has in some instances modified the choice of a mining method; the tendency of massive sulphides to oxidize if broken and left for any considerable time in the stope may result in low recovery by flotation or cause disastrous mine fires.

DESCRIPTION OF EACH MINING METHOD, GIVING ADVANTAGES, DISADVANTAGES, AND CONDITIONS SUITABLE FOR APPLICATION

Open-Stope Method

The open-stope method is one of the most ancient, and was used over 2,000 years ago at certain mines on the Islands of Sardinia and Cyprus, where the Romans mined lead-silver and copper ores, and 1,000 years ago in the Province of Trento in northern Italy. The Mons Argentarius group of silver-lead mines neat Trento, with its many square miles of stoped areas, may still be visited, and one marvels at the great extent of these underground rooms and pillars, excavated by the ancients with chisel and pick.

An open stope is an underground chamber from which the ore has been extracted and in which the roof or walls are supported by pillars of ore or waste at intervals depending upon the strength of the roof or walls, or by props, stulls, or cribbs. This method is used in mining all types of ore deposits, from flat beds to vertical veins.

Underhand stoping or the mill-hole open-stope method, may be used where the roof or walls of the orebody are strong and the ore itself stands well. It is the open-pit bench-m8ning method carried underground. This method is used at the Mascot mine of the

American Zinc Co. in Tennessee and the iron mine (1) of Witherbee Sherman & Co. at Mineville, N. Y. (figs. 1 and 2).

The more general application of the open-stope method embodies attack by overhand stoping, as at the copper mines (20) in Michigan (fig. 3) or by the breast method as in the lead and zinc mines (2, 3, 6, 10, 15) of Tri-State district (fig. 4). Still another variation in open-stope mining is the use of sublevels spaced from 30 to 50 feet apart (11, 12, 17), one above another, and mining the ore into an open chamber at the ends of these sublevels, as indicated in the accompanying sketch. This method is well developed at the Burra Burra mine (5) near Ducktown, Tenn. (fig. 5).

Application.- Open-stope methods are applicable to mining strong or moderately strong orebodies inclosed in strong or moderately strong wall-rocks, some latitude being allowed in this respect depending upon size, shape, and dip of orebody, depth below surface, character of slips and joints in the rock, grade of ore, and the particular variation of the open-stope method employed.

Advantages.- Where the method can be applied, the following are the principal advantages:

1. Low stoping cost per ton, considering that the ore is usually hard.
2. Mechanical loading and haulage methods are easily applied.
3. Ability to follow irregularities in the ore body to secure high total extraction
4. Sorting of waste is possible to some extent and lean ore or barren rock within the ore body may be left in place as pillars.
5. Selective mining of different classes of material is often feasible (16).
6. The ore can be removed as fast as it is broken so that large capital sums are not tied up in broken ore.
7. The sublevel stoping variation of the open-stope method is safe and the accident rate is low (see Disadvantages).
8. Development is usually mostly in ore.

Disadvantages.- The disadvantage of the method are as follows:

1. In some variations of the method considerable ore must be left as pillars to support the roof or walls. Much of this, however, may be recovered later by robbing operations.
2. Except in the sublevel stoping variation, the accident rate is high. (See Table 2.)
3. It is often difficult to inspect and scale the backs and walls.
4. The ground is apt to cave eventually, removing the possibility of mining at some future time adjacent bodies of low-grade ore.

Results.- Two tables follow, one of which shows the results of the open-stope method for mining the different metals, in metal yield and man-hours consumed per ton mined at the mines that produced over $100,000 in 1929; the other gives the details of labor, explosives, and power consumption and costs per ton mined at the individual mines using this method.

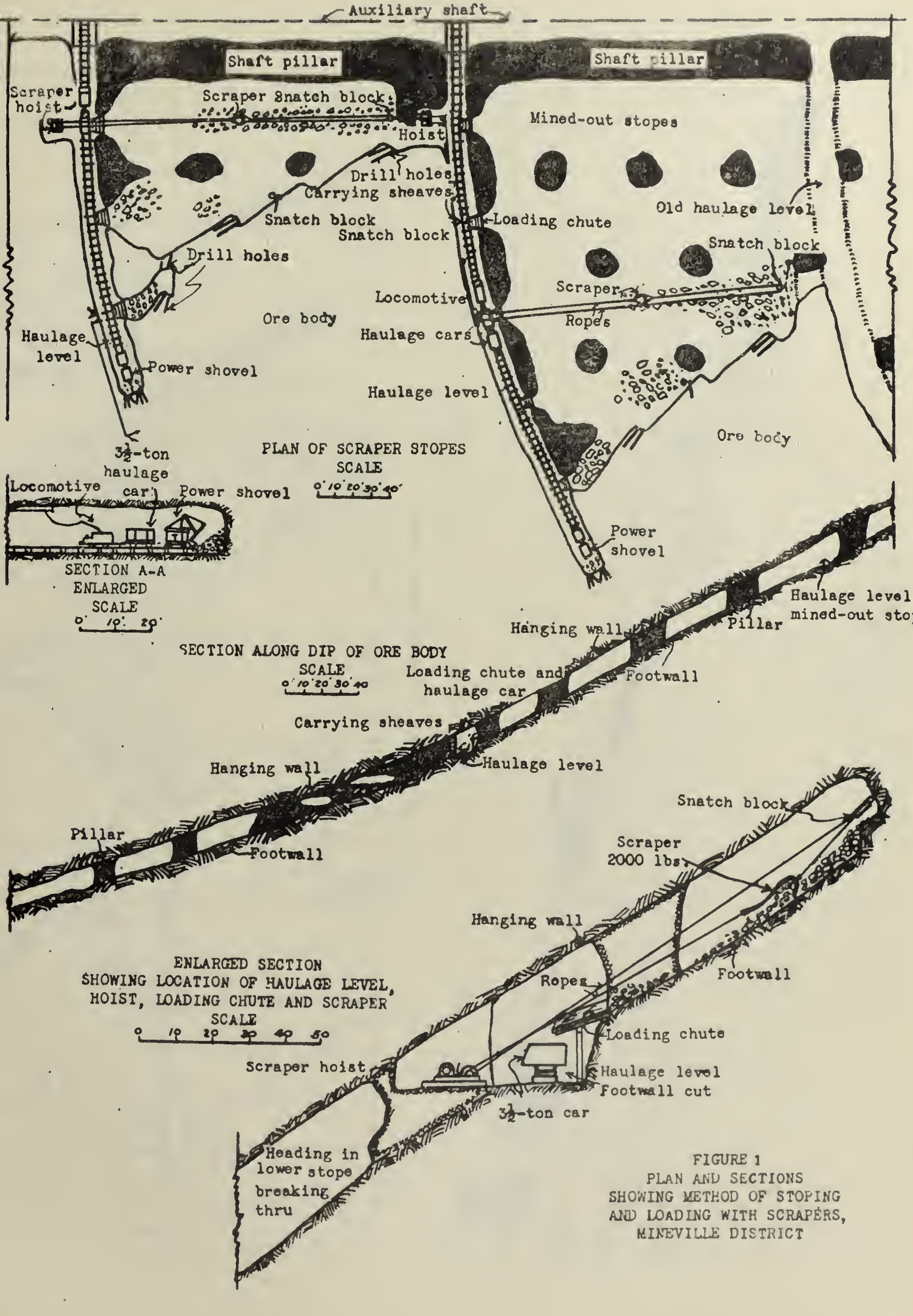
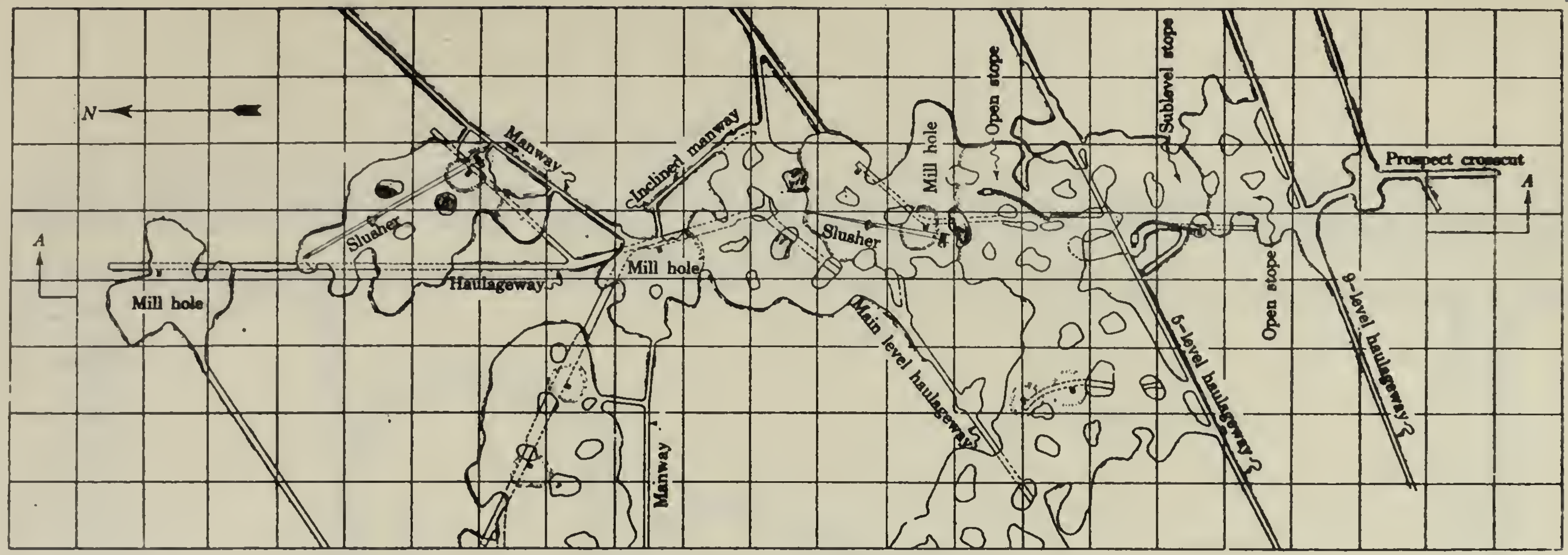
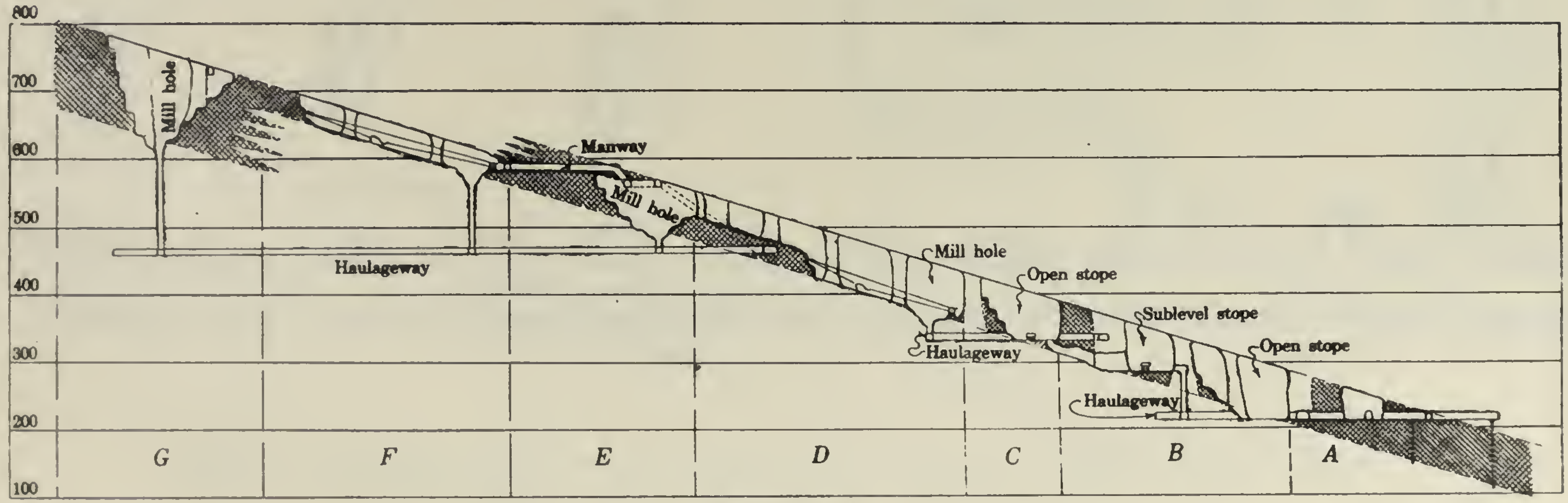


FIGURE 1
PLAN AND SECTIONS
SHOWING METHOD OF STOPING
AND LOADING WITH SCRAPERS,
MINEVILLE DISTRICT



PLAN OF A PORTION OF NO. 2 MINE, MASCOT, TENN.



SECTION A–A

Figure 2.- Open stope mill-hole method, Mascot, Tenn.

0 100 200 300 400 500

Scale in feet

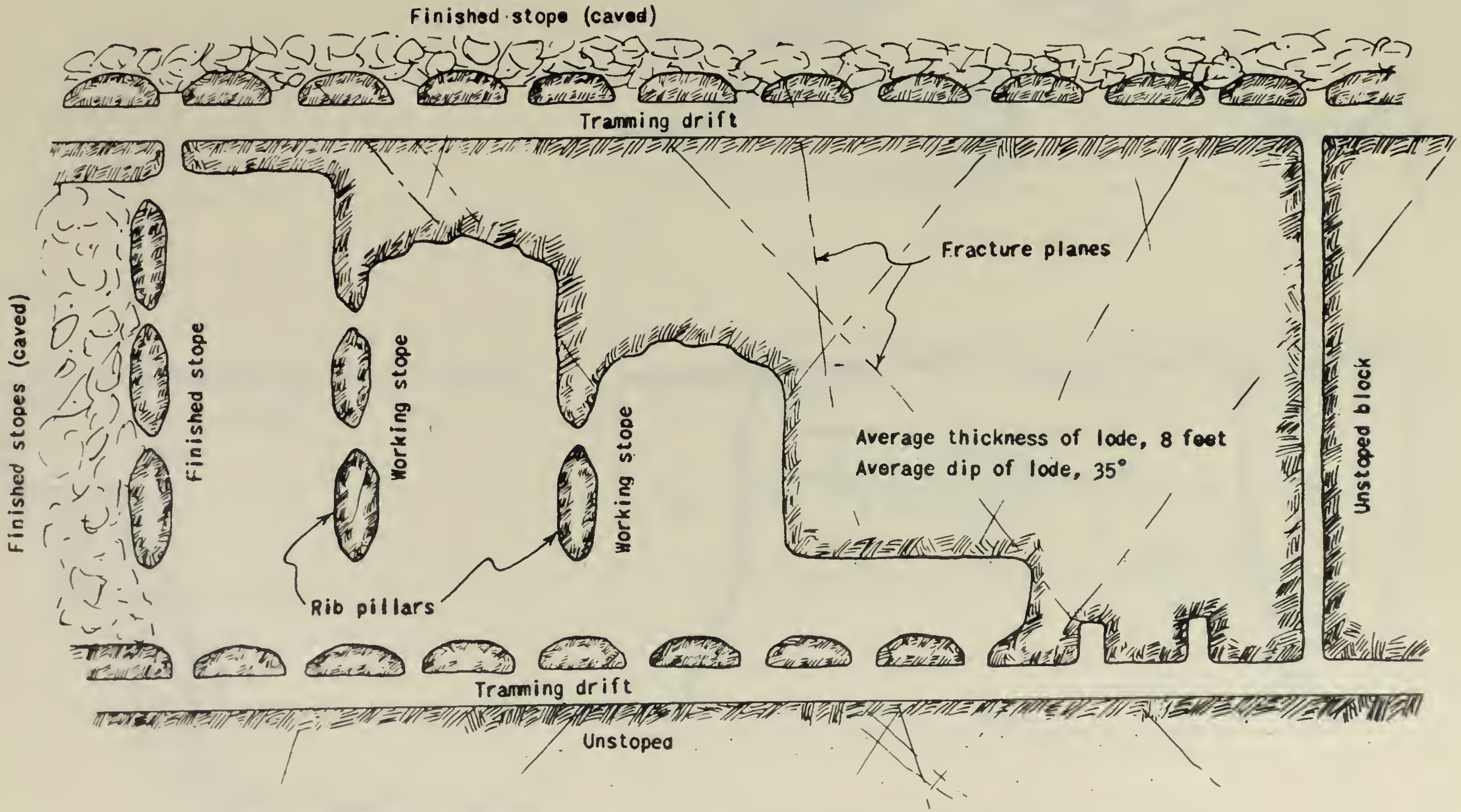


Figure 3.- Retreating system of open stoping with rib pillars.. Section in plane of lode

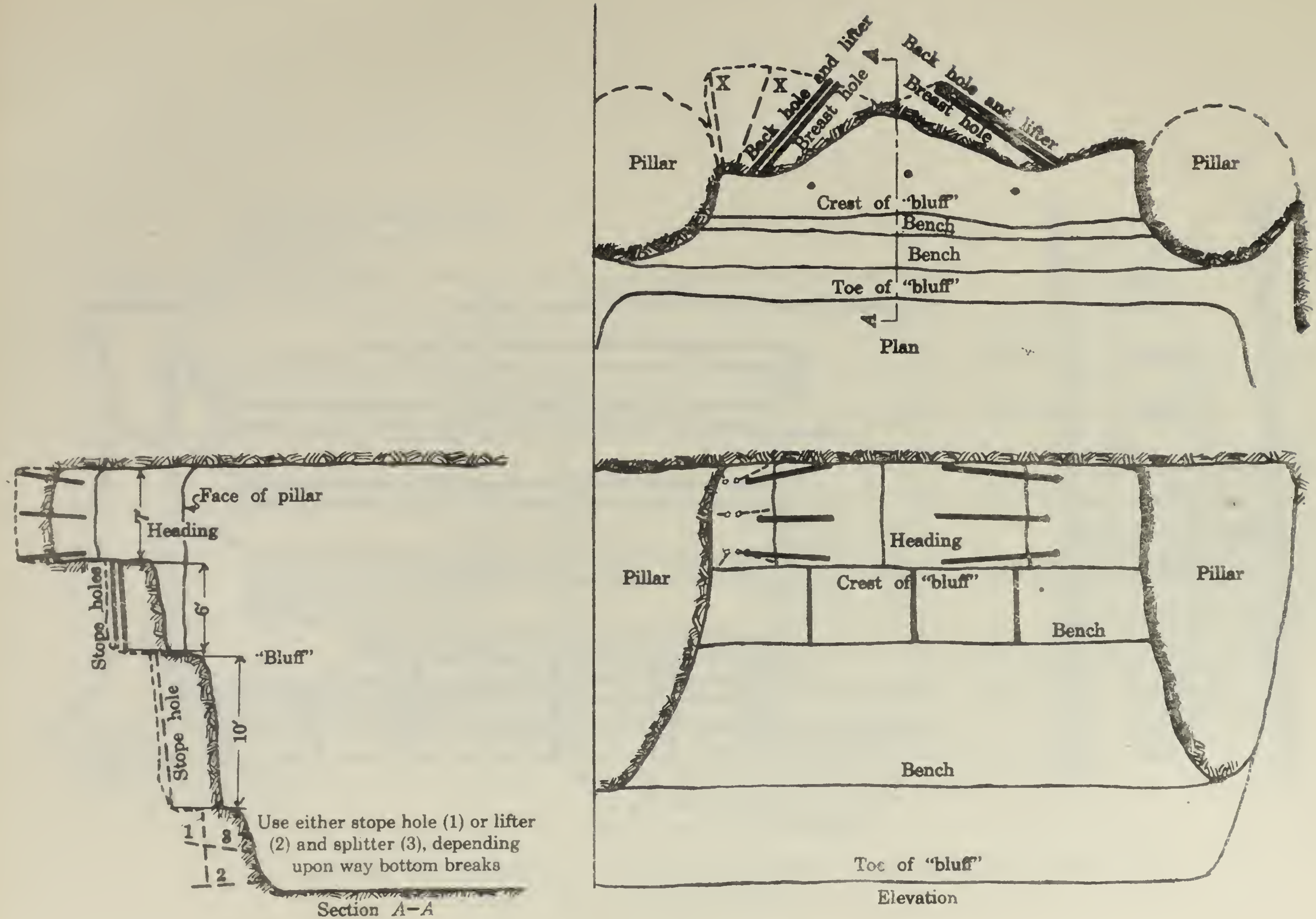


Figure 4.- Heading and stope or "bluff", Southeast Missouri District

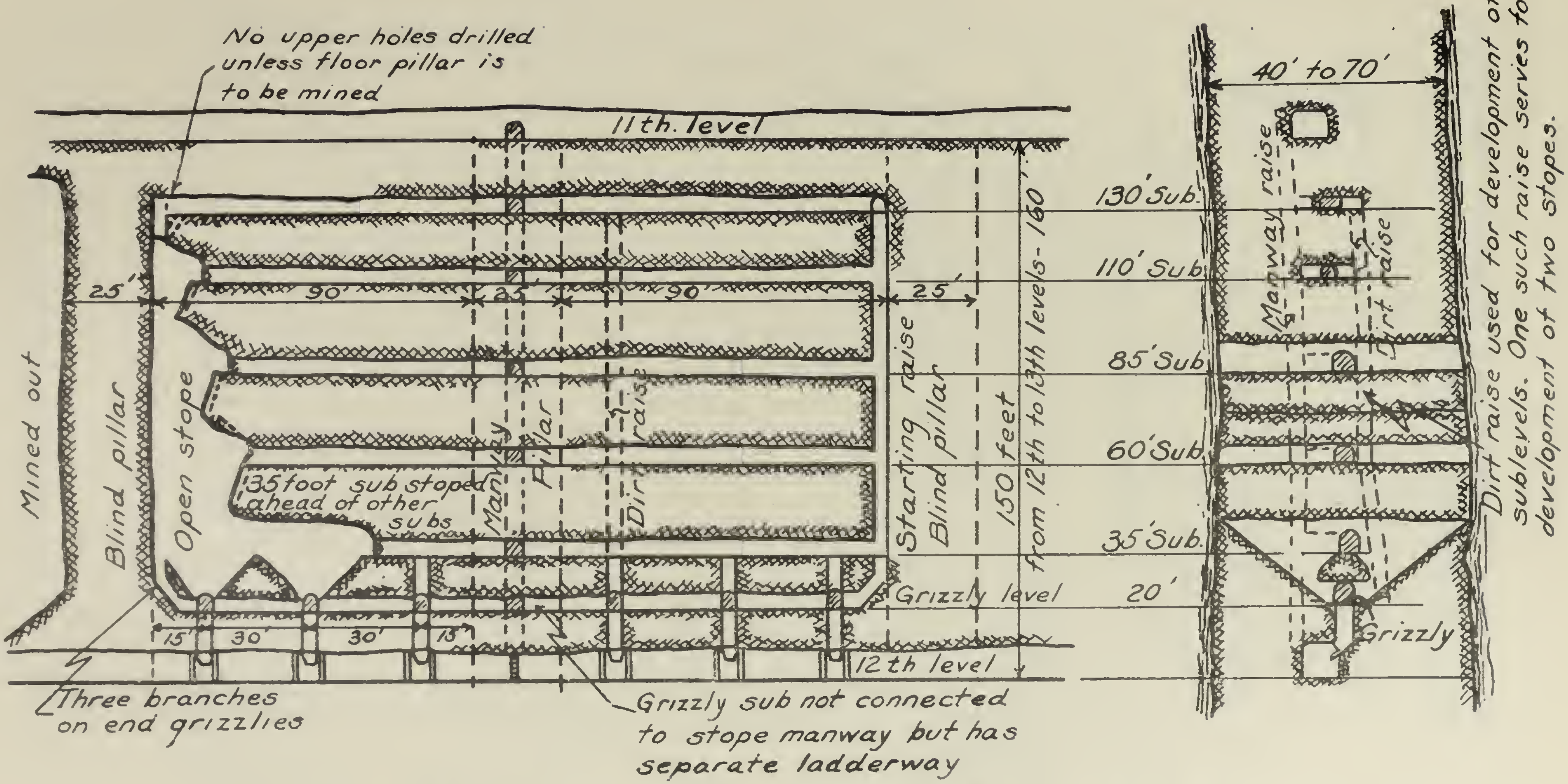


Figure 5. Sublevel stoping in moderately firm and hard ore; stope face overhanging.

Metal yield and man-hours consumed per ton mined by open-stope methods at mines producing over $100,000 in 1929

Metal	Number of mines	Tons mined	Yield per ton	Man-hours per ton
Gold	11	364,516	0.42 oz.	5.54
Copper	11	6,121,545	26.7 lbs	1.37
Lead-zinc	113	15,502,250	68.2 lbs	1.12
Iron	34	11,873,538	.415 ton	1.28

Details of labor, explosives, and power consumption and costs per ton mined at mines using the open-stope method

Mine	Ore	Labor and supply consumption per ton: Labor, total underground, man-hr.	Explosives, pounds	Power, kw.h.	Direct mining costs per ton: Development	Stoping	Haulage and hoisting	General underground expense	Surface applicable to underground	Total	Refer I.C. list, p. 36
FLAT BEDS: 10 TO 200 FEET THICK											
Tri-State No. 1	Pb, Zn	1.341	0.750	9.91	–	$0.630	$0.130	$0.09	$0.135	$0.985	(2)
Tri-State No. 2	Pb, Zn	.876	1.265	5.75	–	.708	–	.165	.120	.993	13)
Tri-State No. 3	Pb, Zn	.836	.875	4.17	–	.645	–	.162	.194	1.001	(10)
Waco	Pb, Zn	1.041	.594	6.65	–	.540)	.155	.057	.033	.785	(6)
Barr	Pb, Zn	.888	.805	6.09	–	.544)	.217	.069	.032	.862	(7)
S. E. Mo. No. 8	Pb	.875	.634	4.56	–	.483)	.222	.135	–	.840	(8)
Mascot	Zn	.831	.502	9.68	$0.053	.222	.278	.086	–	.639	(14)
Hartley-Grantham	Pb, Zn	.745	.741	1.89	–	.516	.196	.049	.021	.782	(15)
Presidio, Texas	Pb, Ag	5.770	2.000	23.70	1.065	2.276	.533	distributed	.127	4.001	(21)
S. E. Mo.	Pb	.827	.550	10.30	–	–	–	–	–	–	(9)
Hanover Bessemer	Fe	.782	–	–	(?)	.306	.167	(?)	(?)	(?)	(16)
BEDS DIPPING OVER 30°; 5 TO 150 FEET THICK											
Marquette No. 1	Fe	1.172	0.901	11.56	0.218	0.770	0.244	0.180	–	1.412	(4)
Burra Burra	Cu	1.242	.755	7.92	.229	.332	.386	.145	0.043	1.135	(5)
Menominee No. 1	Fe	.701	.826	11.80	.211	.467	.173	.052	.027	.930	(12)
Michigan B	Cu	1.220	.991	–	.197	.498	.520	.008	–	1.223	(20)
Michigan C	Cu	1.104	.978	–	.379	.617	.479	.048	–	1.523	(20)
Mary	Cu	2.233	.782	–	.355	.545	.714	.214	–	1.828	(18)
Montreal	Fe	1.008	.850	12.61	.373	.349	.278	.106	.060	1.166	(17)
Mineville, N. Y.	Fe	.958	.937	10.91	.103	.416	.705	.026	–	1.250	(1)
Marquette No. 2	Fe	.919	.422	8.38	.224	.372	.129	.172	.059	.961	(11)
Spring Hill, Montana	Au	1.904	1.262	10.43	.377	1.072	.131	.218	.147	1.945	(19)

Shrinkage Method

Shrinkage stoping was probably first employed for mining fairly thin, tabular ore bodies dipping at steep angles and composed of strong ore with firm walls, and is still the typical method of mining this class of deposit. It is also employed under quite different conditions.

The origin of the method is obscure, but it probably has been used for many years; it might easily have developed naturally in open-stope mining where the back became too high to reach from the bottom of the stope, thus suggesting the use of broken ore as a temporary working floor.

In this method (26 to 36) the ore is stoped upward from the level. The miners stand upon ore broken from preceding cuts, enough ore being drawn off after each cut to leave room to work between the top of the pile of broken ore and the unbroken back (fig. 6, 7, and 8). The pile of broken ore serves two principal purposes: that of a floor upon which to work in attacking the unbroken ore above; and that of a temporary support to the walls of the stope.

Application.- Shrinkage stoping is applicable to the mining of bodies of ore firm and strong enough to be self-supporting over considerable spans and with firm wall rocks. It finds its best application in the mining of narrow vein deposits dipping over 45°. Although it is often employed for mining wide ore bodies, it is probable that in many such cases other methods, such as sublevel stoping or modified shrinkage, employing large blasts (26) would be superior (fig. 9).

This method also has an important application in conjunction with other stoping methods, such as with block caving in boundary cut-off stopes.

Advantages (30).-

1. For mining bodies of strong ore with firm wall rocks the shrinkage method quickly develops and brings the mine into the productive stage, as only a small amount of preliminary development and stope preparation are required. This is particularly advantageous to small companies with limited working capital. (See par. 1, under Disadvantages.)

2. The broken ore in the stopes assists in supporting the walls and eliminates the use of timber for this purpose except as props and stulls may be required to support occasional loose ground.

3. The miners work on a solid floor which provides a sure footing making that feeling of safety and security conducive to efficient work.

4. Rapid working of the stope is possible. A large number of men can often work to advantage in a single stope.

5. Long ore passes extending from level to level are not required, since the ore is drawn from the bottom through a number of chutes or short raises.

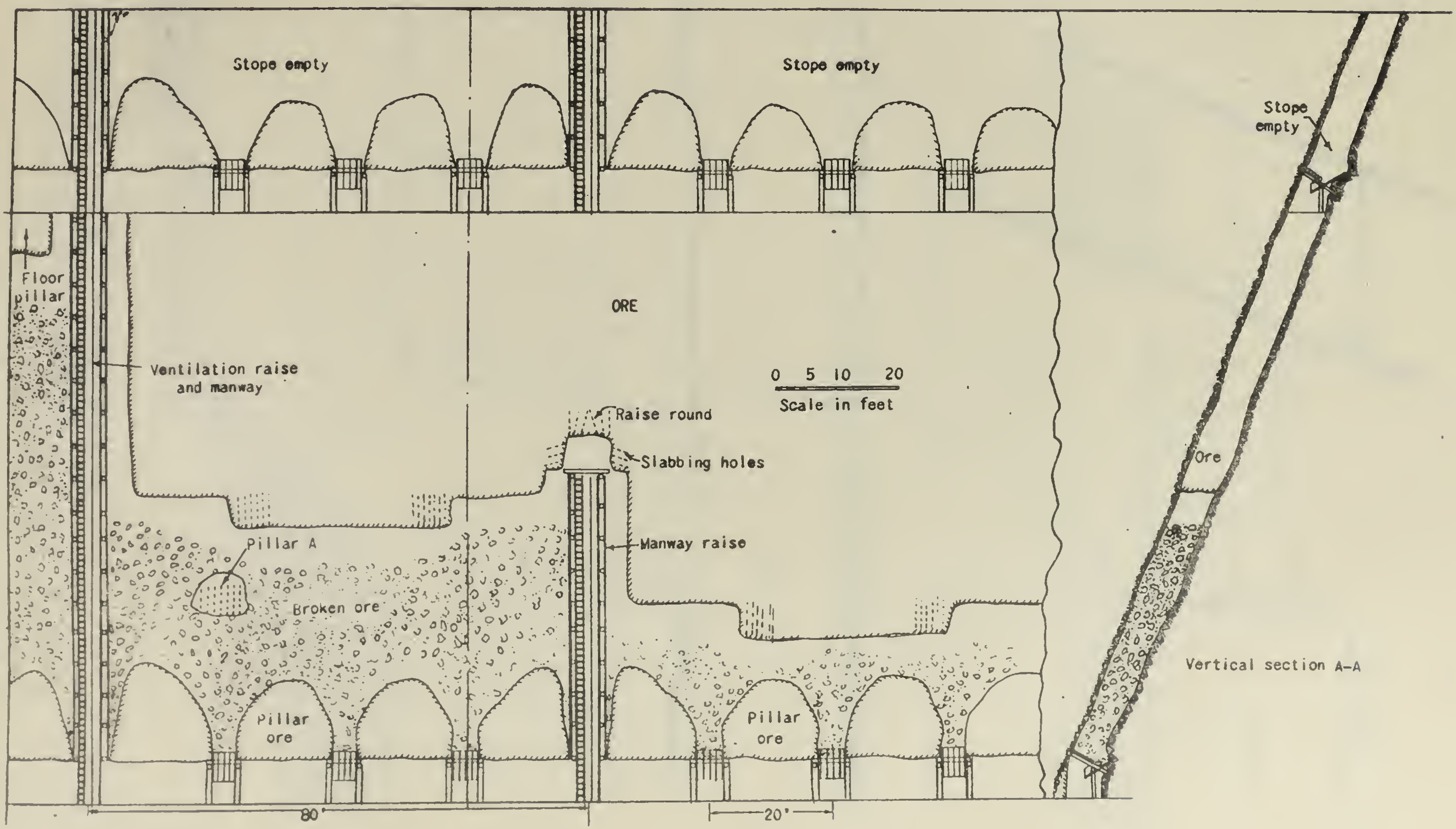
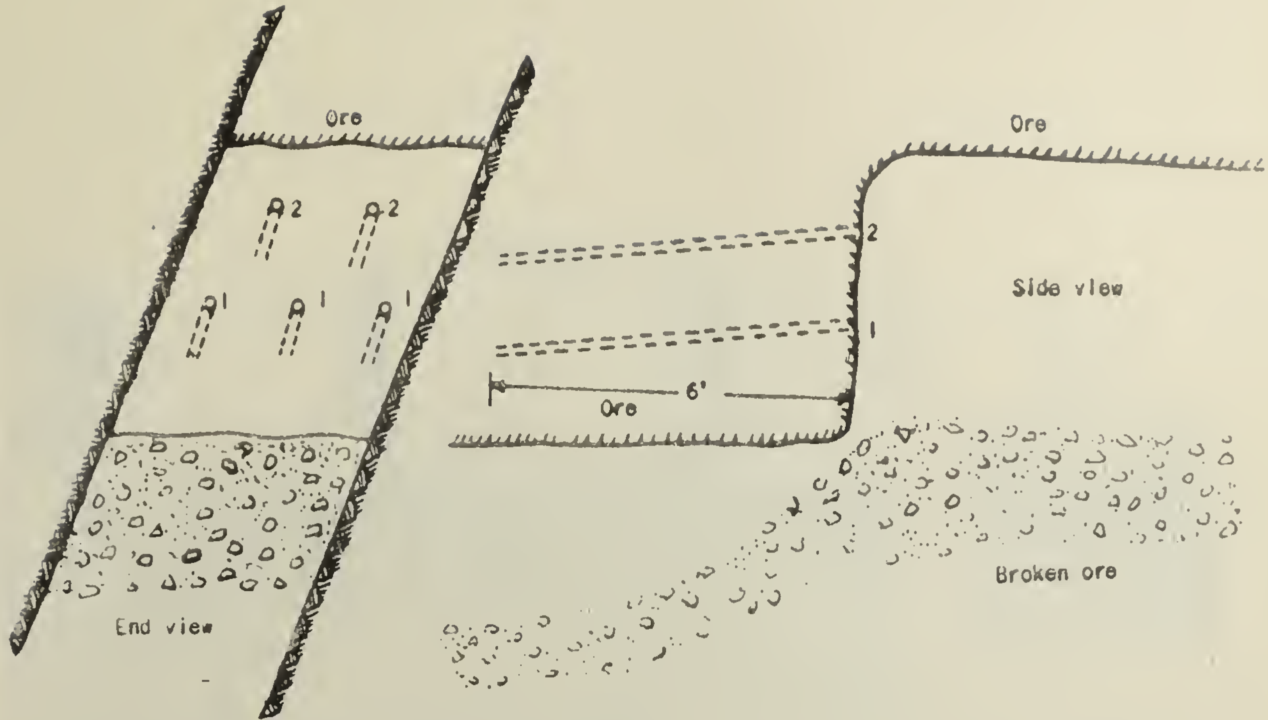
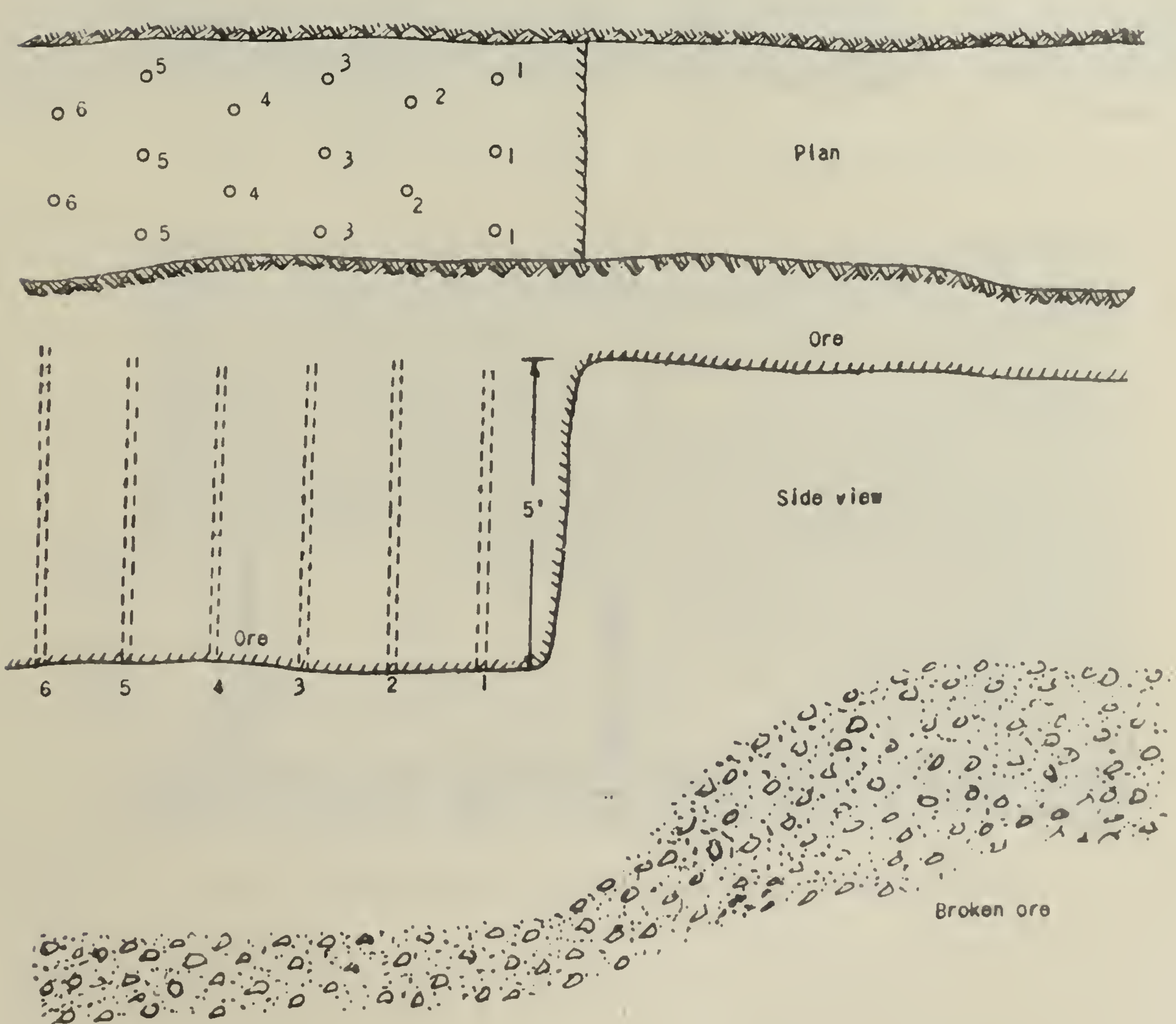


Figure 6.– Longitudinal and vertical section of shrinkage stope



Five-hole stope round in hard ore.



Fifteen-hole stope round in soft ore

Figure 7 - Stope rounds

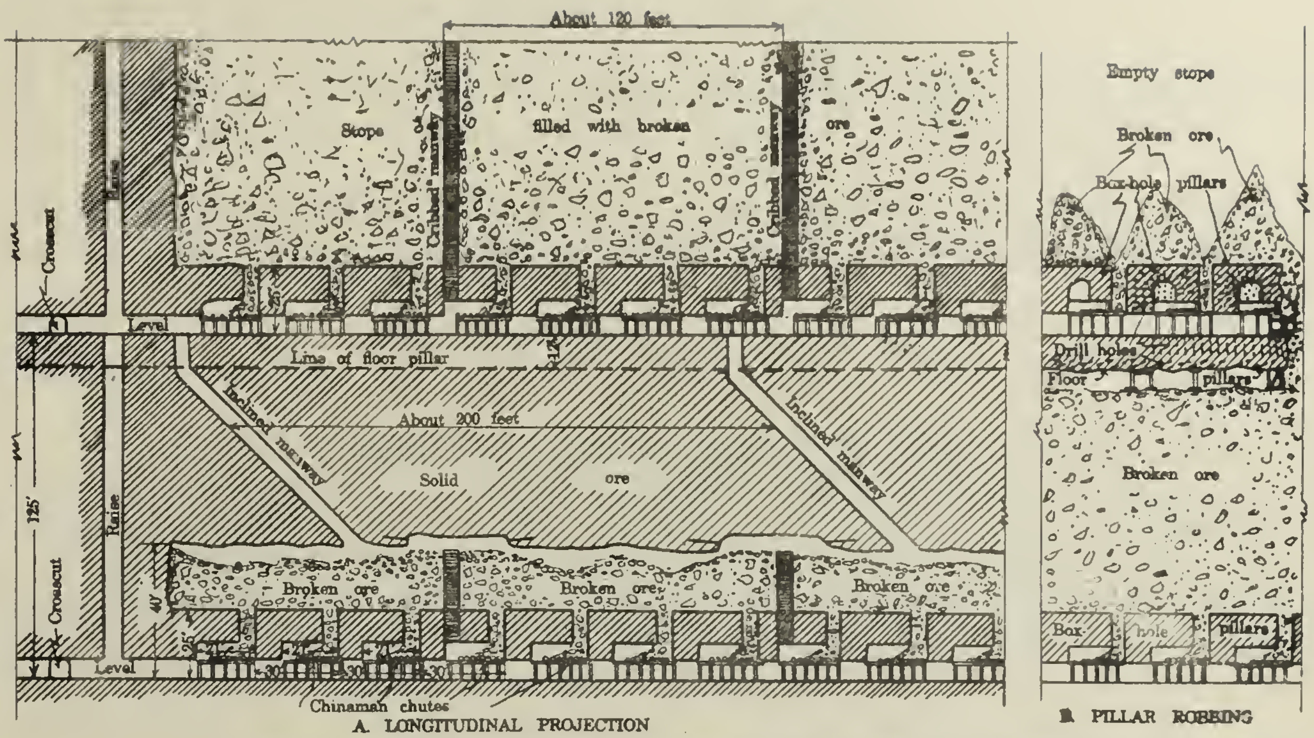


Figure 8. – Shrinkage stoping over box-hole pillars, using chinaman chutes, at Kirkland Lake, Ontario

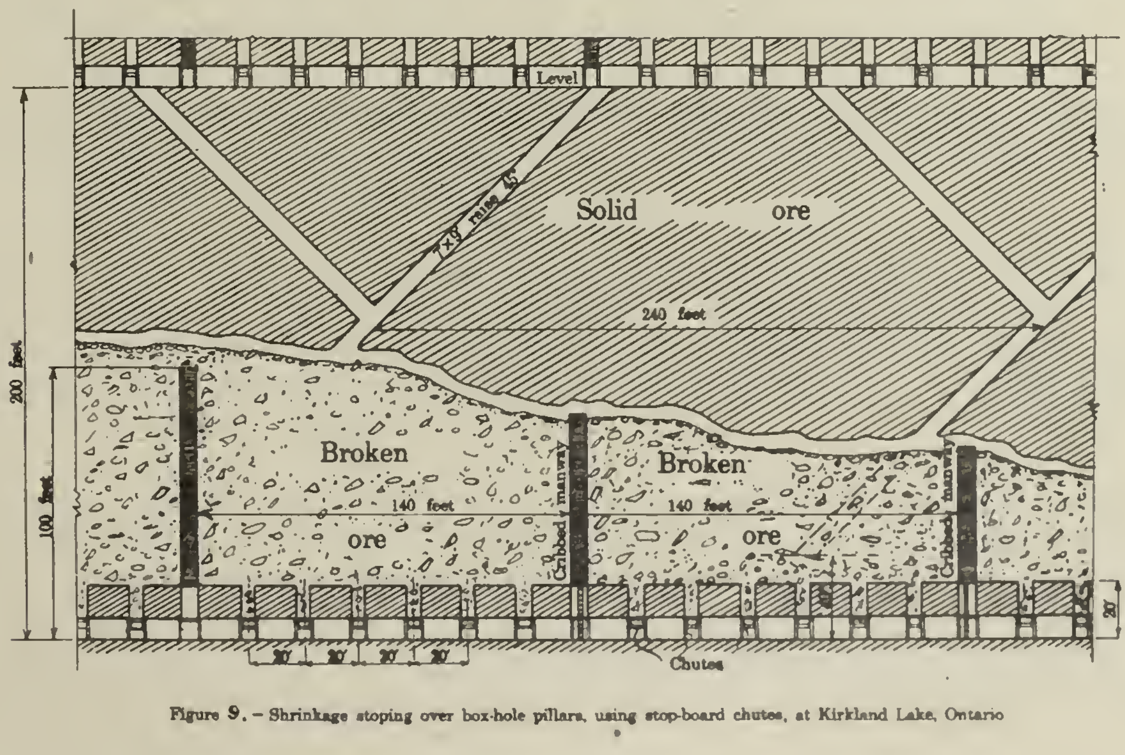


Figure 9. – Shrinkage stoping over box-hole pillars, using stop-board chutes, at Kirkland Lake, Ontario

6. Tramming or wheeling ore in the stopes is not required.

7. A large reserve of ore is maintained. (See par. 4, under Disadvantages.)

8. Good ventilation can be obtained at little expense.

9. Large blocks of ore can be conveniently broken up in the stopes, thus to a large extent eliminating blasting in the chutes. (See par. 7, under Disadvantages.)

10. The method is cheaper on a per ton of ore basis than cut-and-fill stoping, but not necessarily so on a per unit of metal basis. (See U. S. Bureau of Mines Bull. 306, p. 187.)

Disadvantages.-

1. Although the shrinkage method permits rapid development of a mine to the stoping stage, only about 35 per cent of the ore broken in a given stope is immediately available, the balance being tied up in the stope till it is worked out. For companies having only small working capital, this situation requires that a considerable portion of that capital be expended for advance development work and for breaking ore which can not be turned into cash for a considerable period of time. This disadvantage is particularly important in the case of high-grade ore. (See par. 1, under Advantages.)

2. While the broken ore serves as a support for the walls, if the walls are friable or scaly, considerable contamination of the ore with wall rock may result while the broken ore is drawn, both during stoping and when finally emptying the stope.

3. Though long ore passes are not required, chutes must be more closely spaced than is required in some other methods applicable to the same type of orebodies in order that the broken ore may be drawn down evenly during operation.

4. This method affords little opportunity for sorting lean ore and waste in the stopes. Irregular masses of ore or offsheets from the main vein may be lost. This is especially true in the footwall because these masses are covered up with broken ore or because of the difficulty of disposing of waste which would be broken in following stringers that may enlarge into sizeable bodies in the wall.

5. Manways through broken ore are sometimes difficult to maintain owing to the drag of the ore during drawing, especially in wide veins. This may necessitate driving raises in the wall rocks or in ore pillars for access to the stopes and for ventilation.

6. Although large blocks of ore can be conveniently broken up in the stope (par. 9, under Advantages), large chunks may become covered with dirt and fines and may not be discovered until they give trouble at the chutes.

7. Certain pyritic ores will take fire if left broken in stopes. More often only slight oxidation occurs, but frequently even slight filming of sulphides will result in greatly reduced recovery of the valuable minerals by flotation and may be a serious enough factor to preclude the use of the shrinkage method.

8. The necessity in some cases of filling the stopes after drawing the broken ore in order to prevent caving of the surface or of overlying veins is common to other stoping methods. As compared to the cut-and-fill method, however, delayed filling after emptying the shrinkage stopes should be cheaper, since it does not usually require that definite amounts of filling material be provided at definite times in order to accommodate stoping operations. With cut-and-fill methods, filling, usually in comparatively small quantities at a time for any given stope, is required promptly and at frequent intervals when the filling cycle of the stoping operation is reached.

9. Travel in shrinkage stopes is often difficult, sometimes being up and down hill between and over draw points. Much time is consumed in moving drills and gear about.

10. Accidents due to sudden subsidence of broken ore sometimes occur in narrow veins where the broken ore hangs up. They can be avoided, however, by proper cooperation between stope bosses and men in opposite shifts.

Results.- Two tables follow, one of which shows results obtained by the shrinkage method for the different metals, in metal yield and man-hours consumed per ton mined at the mines producing over $100,000 in 1929; the other table gives the details of labor, explosives, and power consumption and costs per ton mined at individual mines using this method.

Metal yield and man-hours consumed per ton mined by the shrinkage method at mines producing over $100,000 in 1929

Metal	Number of mines	Tons mined	Yield per ton	Man-hours per ton
Gold	11	5,645,110	0.17 oz.	0.69
Copper	12	2,306,930	33.6 lbs	2.45
Lead-zinc	10	2,011,500	144.4 lbs	2.04
Iron	4	955,900	.456 ton	1.04

Details of labor, explosives, and power consumption and costs per ton mined at mines using shrinkage-stoping methods

Stoping method	Mine	Ore	Width and dip of orebodies	Labor and supply consumption per ton: Labor, total underground, man-hr.	Explosives, pounds	Timber	Power, kw.h.	Direct mining costs per ton: Development	Stoping	Haulage and hoisting	General underground expense	Surface applicable to underground	Total	Refer I.C. list, p. 36
Shrinkage	Nevada-Massachusetts	W	Av. 4½ ft.; 75°	3.051	1.789	3.63 bd. ft. 0.61 lin. ft.	10.48	$0.907	$2.743	$0.190	$0.105	-	$3.945	(29)
Shrinkage	Black Butte	Hg	15 to 70 ft.; 58°	1.713	1.160	-	18.45	-	-	-	-	-	1.310	(28)
Shrinkage	Teck Hughes	Au	0 to 60 ft.; 75° av.	2.497	2.360	3.52 bd. ft.	19.56	.433	1.557	.443	.202	$0.435	3.070	(32)
Shrinkage	Cortez	Ag	1 to 20 ft.; 45 to 80°	3.362	3.300	2.27 bd. ft.	21.18	1.427	1.972	.467	-	-	3.866	(33)
Shrinkage	Engels	Cu	100 ft.; 80°	1.535	2.758	1.23 bd. ft. 0.074 lin. ft.	9.64	.479	.760	.438	-	-	1.677	(27)
Shrinkage	Michigan D	Cu	3.5 to 20 ft.; 56°	1.239	.983	0.805 lin. ft.	-	.218	.482	.556	.240	-	1.496	(20)
Shrinkage	Kirkland No. 2	Au	8 to 40 ft.; 75 to 85°	1.858	1.624	1.67 bd. ft. 0.59 lin. ft.,	-	1.450	2.210	.830	-	-	4.490	
											-	-	-	
Shrinkage	Kirkland No. 4	Au	3 to 20 ft.; av. 60°	2.810	2.270	0.864 lin. ft.	23.00	-	-	-	-	-	-	
Shrinkage	Porcupine No. 1 =	Au	4 to 40 ft.; 45 to 90°	2.740	2.210	0.42 bd. ft. 0.70 lin. ft.	32.22	.790	-	-	-	-	-	
Shrinkage	Porcupine No. 2	Au	6 to 60 ft.; steep dips	2.700	1.277	1.39 lin. ft.	20.81	.850	(2.270		.200	.090	3.210	
Shrinkage, square-set, and caving	Homestake	Au	25 to 250 ft.; 70°	1.161	.753	4.87 bd. ft.	10.00	.130	1.143	.320	.191	-	1.784	
Modified shrinkage and caving	Alaska-Juneau	Au	Wide	.266	.400	-	1.61	.061	.122	.114	-	-	.297	(26)
Shrinkage with pillar caving	Michigan Iron	Fe	Max. 140 ft.: 70°	.595	.501	3.24 bd. ft. 0.544 lin. ft.	4.36	-	-	-	-			
Semishrinkage	Marquette No. 4	Fe	5 to 40 ft. thick; 30 to 35°	1.130	.904	0.006 lin. ft.	12.34	.212	.801	.205	.053	.009	1.280	(35)
Shrinkage; some cut-and-fill	Eighty-Five, New Mexico	Cu	2 to 10 ft. thick; av. 80°	6.230	-	-	96.25	-	1.970	1.140	1.170	.210	4.490	(36)
Shrinkage	Verde Central	Cu	5 to 100 ft. thick	1.670	1.160	3.08 bd. ft.	16.44	-	1.661 incl. stope development	.580	.111	.033	2.385 development excluded	(37)

Cut-and-Fill Method

Cut-and-fill stoping has long been used in many parts of the world to mine bodies of ore with too weak to stand for any length of time unsupported and where it has been necessary or desirable to prevent caving of the ground and subsidence of the surface. It has been widely used in Europe for many years.

By this method (45 to 51) the ore is stoped upward from the level by a series of flat or inclined slices (figs. 10, 11, and 12). After each slice is taken, the broken ore is removed and waste filling is run in to within a few feet of the solid back. The miners then stand on this waste material in making the next cut. A regular cycle of operations is carried out: Breaking; removal of broken ore; and filling; and then the cycle is repeated.

Application.- Cut-and-fill stoping is usually applied to the mining of medium to high grade vein or bedded deposits of moderately strong to strong ore having dips of 55° or more and where one or both walls are relatively weak.

The ore must stand well over moderate spans for a short time. This method is also applied to wide, thick bodies of hard, high-grade ore by mining stopes separated by pillars which are later removed by the same or other methods. It is also used instead of shrinkage stoping where it is desired to extract the ore at once rather than to leave it in the stopes for the considerable periods necessitated by the use of the shrinkage method.

Advantages.-

1. Bad walls may be safely held and the general stability of the mine maintained by the filling.

2. Irregular ore bodies can be easily and completely mined and offshoots or tongues of ore in the walls can be followed, leaving broken waste in the stope. The ore body may be prospected as stoped to some extent.

3. Horses of lean or barren rock can be left unbroken and waste can be sorted out from the broken ore and left in the stope.

4. There is less dilution than with shrinkage stoping.

5. Secondary breaking (bulldozing) can be done in the stope, and all material put in the chutes can thus be prepared for easy drawing and handling.

6. The accident rate is lower than for shrinkage stoping and square-setting.

7. The broken ore is removed as fast as broken, so that large capital sums need not be tied up in broken ore reserves.

8. Stopes are easily ventilated.

Disadvantages.-

1. The rate of output from the stope is limited or else is irregular due to stopping the breaking of ore during filling operations.

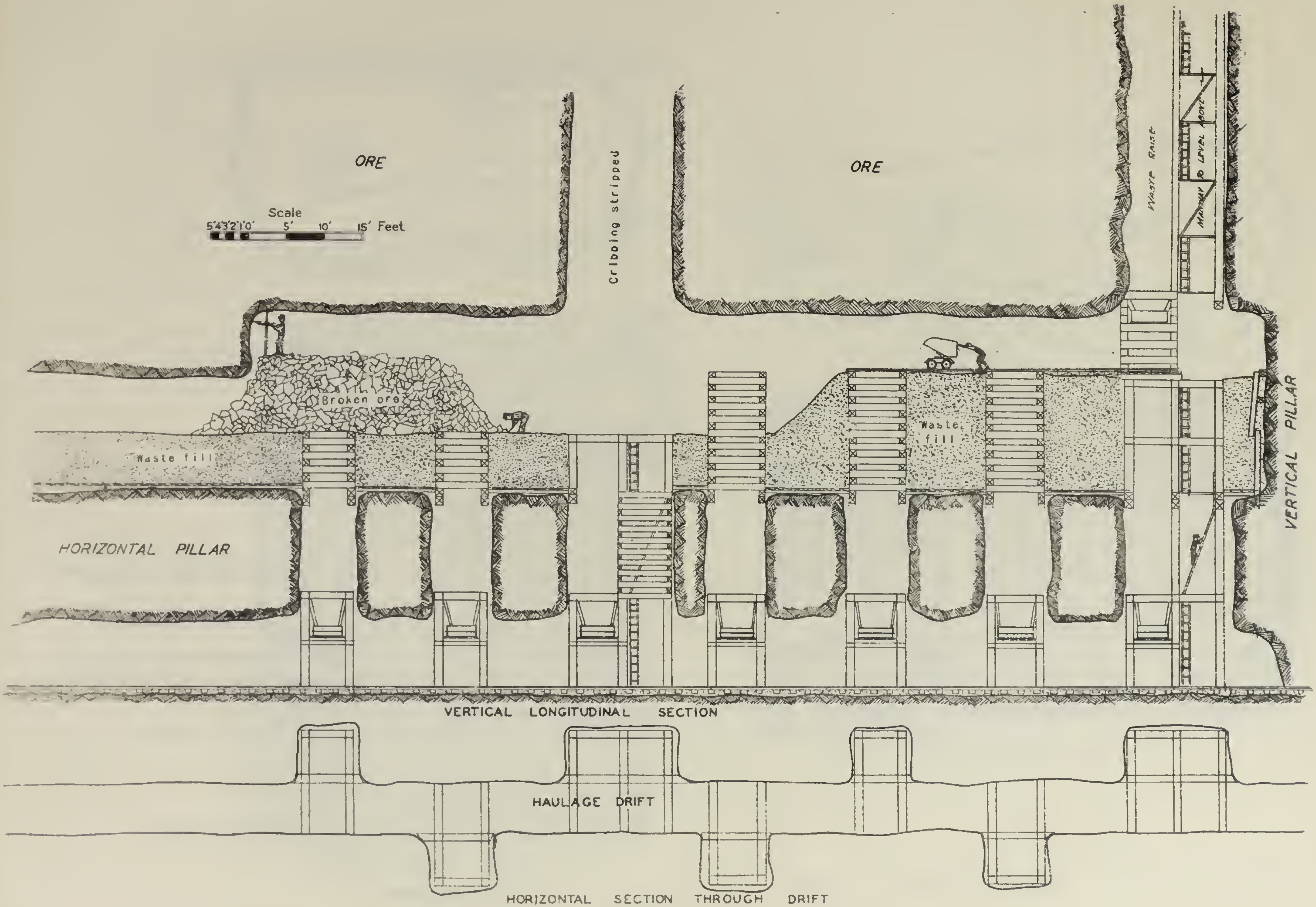


Figure 10 - Typical cut-and-fill stope

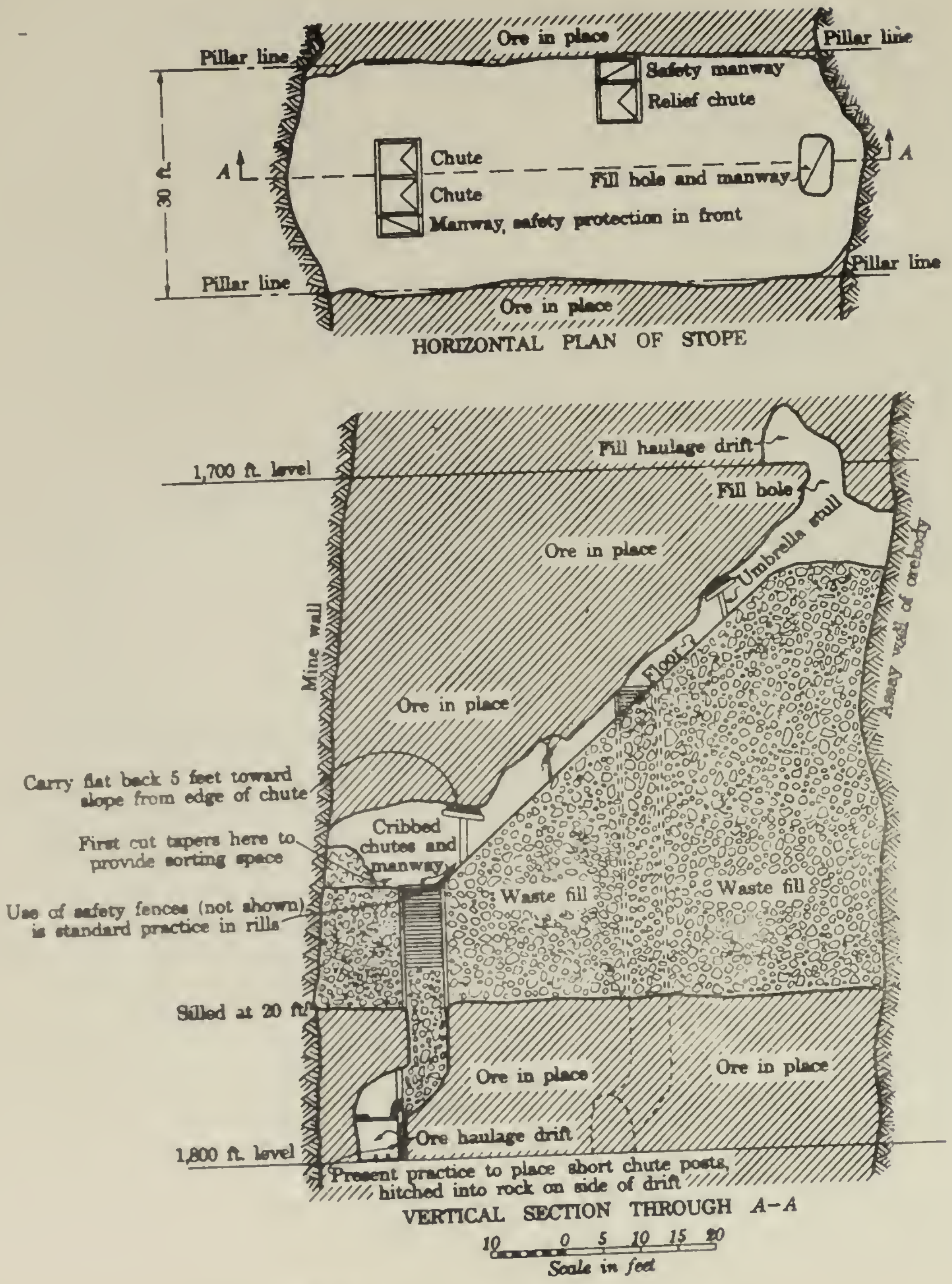


Figure 11.- Typical rill section

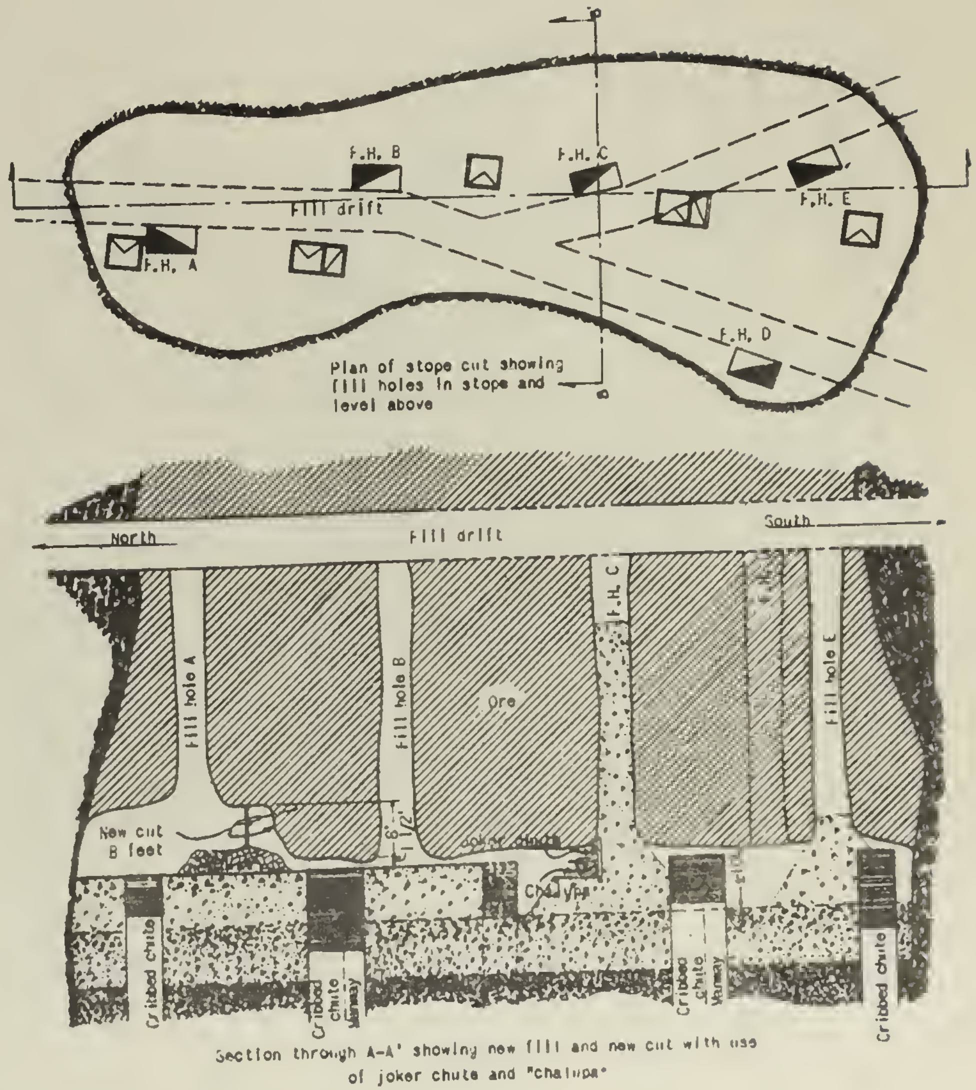


Figure 12.- Typical cut-and-fill slope in operation

2. Filling must be provided at such times and in such quantities as are required to maintain a regular cycle of stope operations.

3. Cost is higher per ton of ore mined than for shrinkage stoping but is usually lower than for square-setting.

4. Handling cost of ore and waste in stopes is often high, or else ore and waste passes are installed at such close intervals that their cost is great.

Results.- Two tables are given; one shows the different metals, the metal yield and man-hours consumed per ton mined by the cut-and-fill method at mines that produced over $100,000 in 1929; the other gives details of labor, explosives, and power consumption and costs per ton mined at individual mines using this method.

Metal yield and man-hours consumed per ton mined by cut-and-fill methods at mines producing over $100,000 in 1929

Metal	Number of mines	Tons mined	Yield per ton	Man-hours per ton
Gold	2	13.150	1.03 oz.	15.78
Copper	6	1,573,135	67.5 lbs.	4.18
Iron	2	272,835	.550 ton	2.25
Lead-zinc	11	664,252	161.72 lbs.	5.48

Details of labor, explosives, and power consumption and costs per ton mined at mines using cut-and-fill methods

Stoping method	Mine	Ore	Width and dip of orebodies	Labor and supply consumption per ton: Labor, Total underground, man-hr.	Explosives, pounds	Timber	Power, kw.h.	Direct mining costs per ton: Development	Stoping	Haulage and hoisting	General underground expense	Surface applicable to underground	Total	Refer I.C. list, p. 36
Cut-and-fill	Matahambre	Cu	20 to 30 ft.; 42 to 45°	4.310	1.020	1.146 bd. ft. 0.148 pcs. props	14.01	$0.711	$1.267	$0.388	$0.184	$0.019	$2.569	(45)
Cut-and-fill, some square-sets, top slicing and shrinkage	Pilares	Cu	Various: steep	3.790	.383	5.71 lin. ft.	6.20	.300	1.260	Included with development and mining	1.110	-	2.670	(48)
Cut-and-fill, and square-set	Pecos	Pb, Zn Cu	Narrow stringers to 40 ft.; nearly 90°	5.674	.730	3.90 lin. ft.	11.78	1.218	2.396	.711	.563	.010	4.898	(49)
Cut-and-fill	Michigan E	Cu	3.5 to 20 ft.; 70°	3.625	.647	3.22 bd. ft.	8.19, plus 70 lbs. coal	.598 (haulage included)	1.765	.158 hoisting only	-	-	2.699	(20)
Inclined cut-and-fill, and semishrinkage	Campbell (C & A)	Cu	50 to 250 ft.; 25 to 90°	-	$0.12 per ton	$0.175 per ton	-	-	1.280	-	-	-	-	(47)
Cut-and-fill, 64% Square-set, 27% Top slicing, 8% Shrinkage, 1%	United Verde	Cu	Large	2.7 approx.	.55	6.95 bd. ft.	-	-	1.148	.238	1.037	.122	2.544	(50)
Cut-and-fill	Block P Mine	Pb, Zn	1 to 4 ft.; 65 to 88°	5.350	2.850	3.95 bd. ft. 1.30 lin. ft.	17.45	-	-	-	-	-	-	(51)

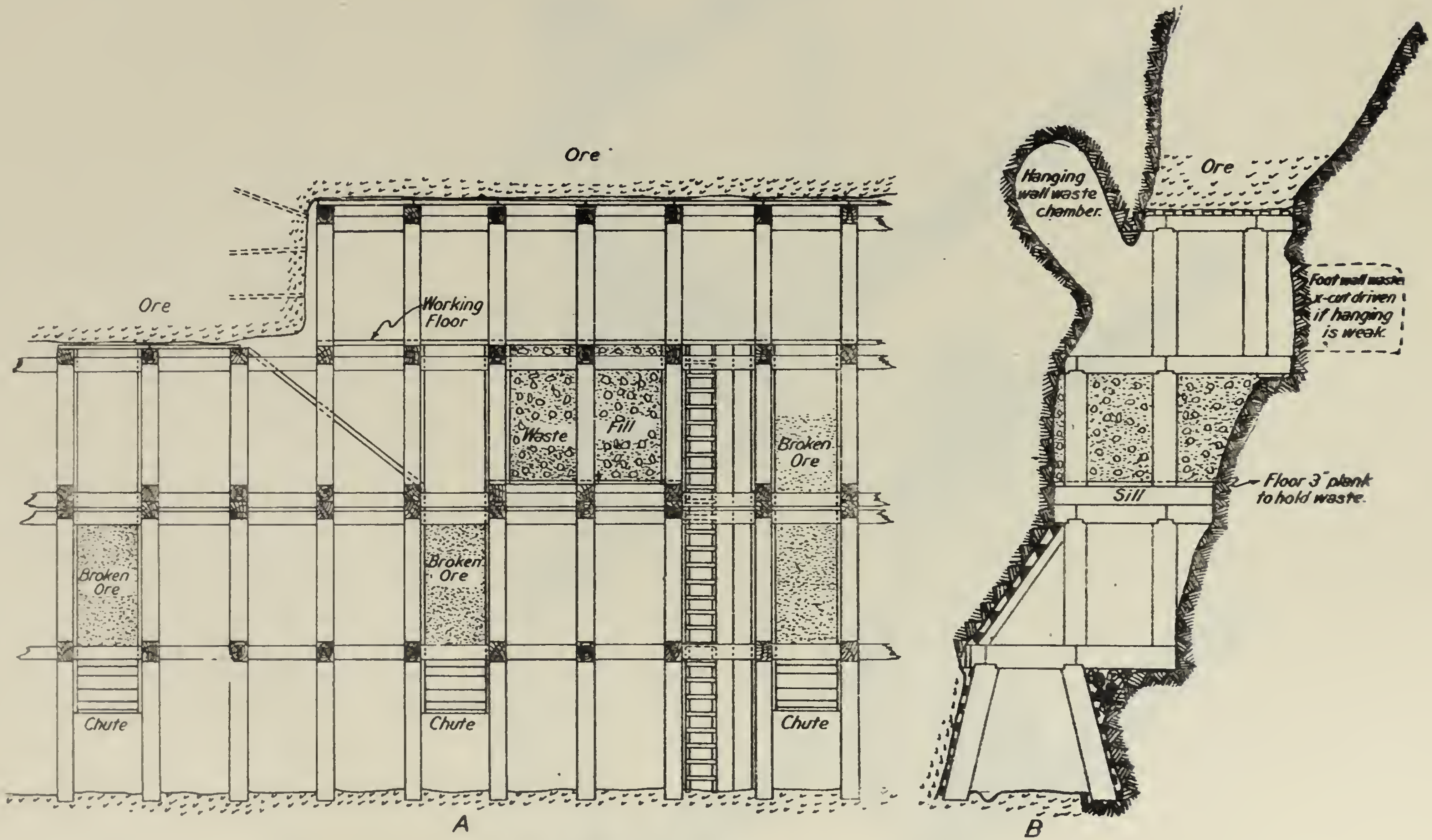


Figure 13 - Vertical sections of stope.
A: Longitudinal section. B: Cross-section.

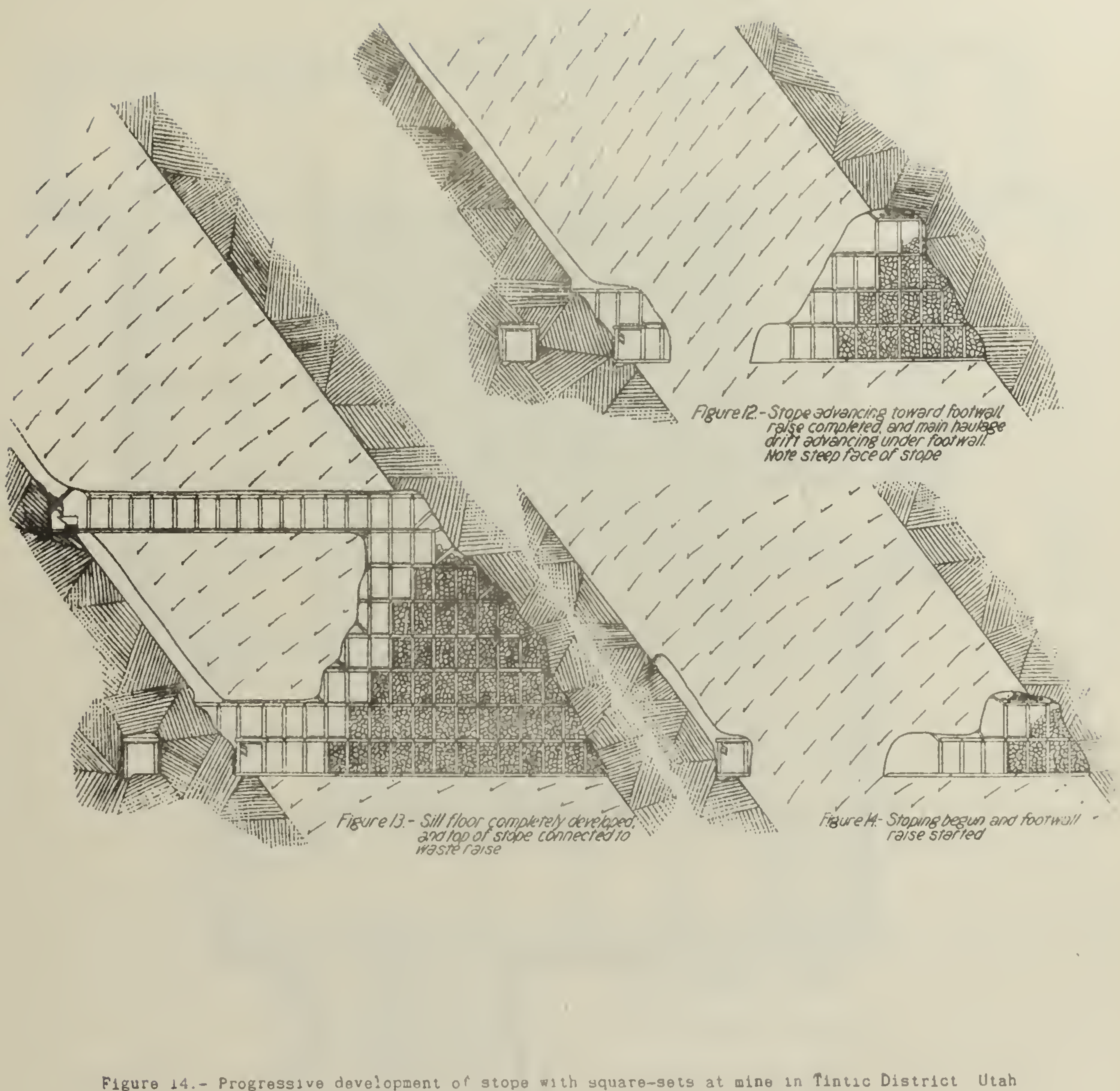


Figure 14.- Progressive development of stope with square-sets at mine in Tintic District Utah

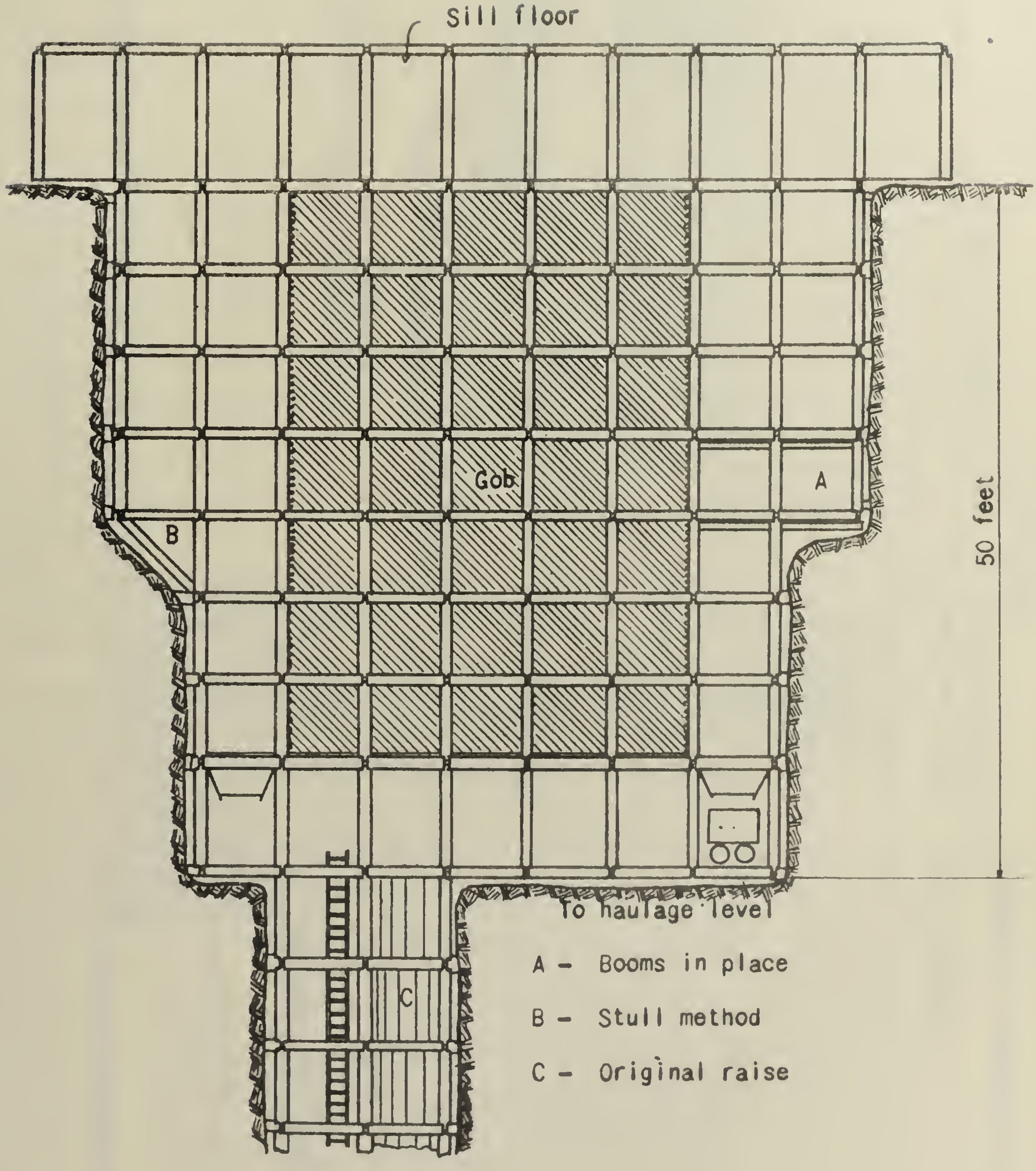
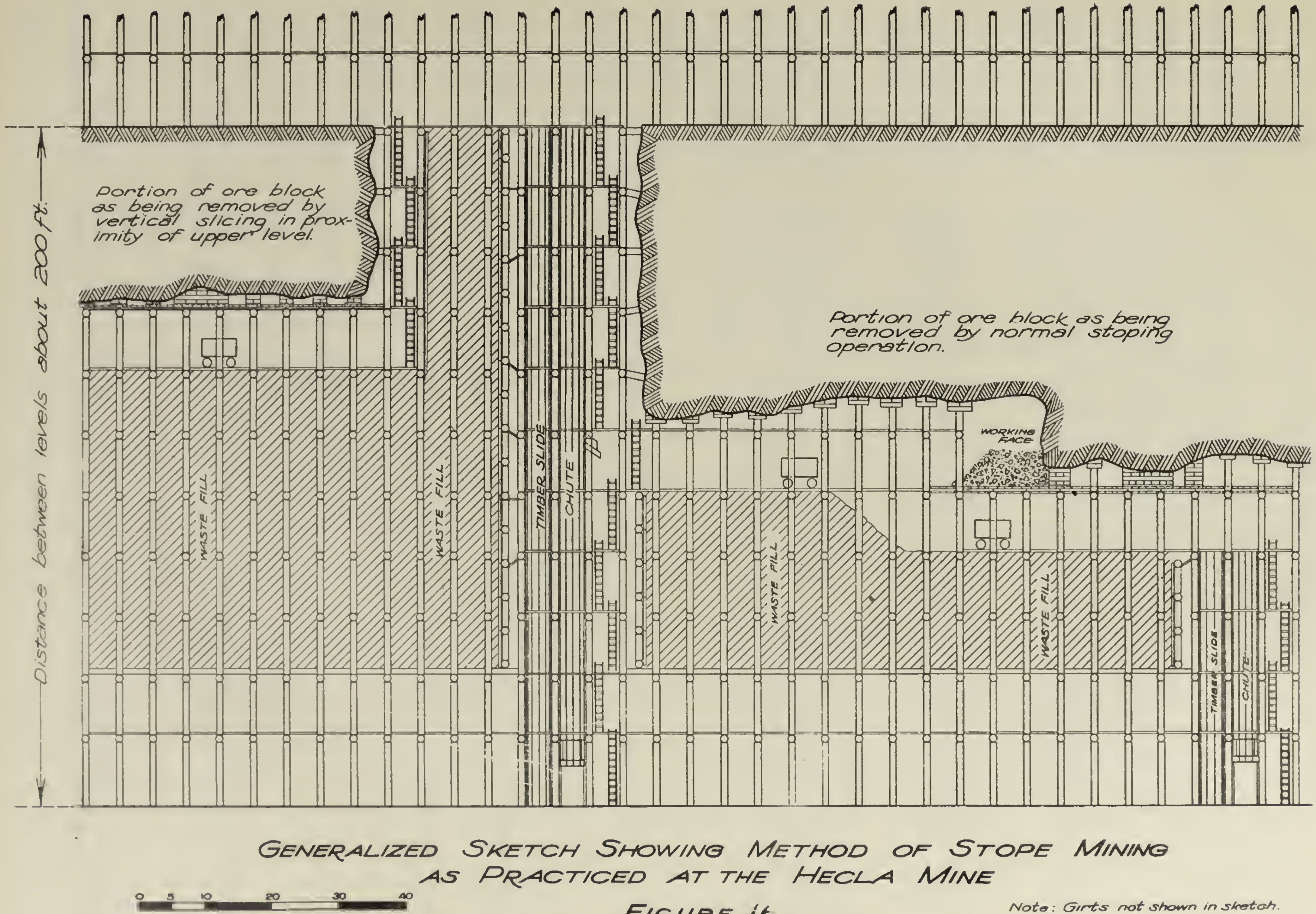


Figure 15.- Section showing underhand stopes



GENERALIZED SKETCH SHOWING METHOD OF STOPE MINING AS PRACTICED AT THE HECLA MINE

FIGURE 1t

Note: Girts not shown in sketch.

Square-Set and Stull-Set Stoping Methods

Square-set stoping has been employed for many years and in widely separated districts to mine regular and irregular deposits of ore of appreciable vertical dimension where the ore and walls will not stand without immediate support except over very small spans and where the back and walls must be artificially supported against caving. Some engineers claim that the square set method was first used on the Comstock Lode. In this method the ore is excavated in small, rectangular blocks, one block at a time. After each block is removed, its place is taken by a set of framed timber which serves as a temporary support to the surrounding ground. As best employed to-day, square-set timbering is not used to permanently support the stope nor can it hold great pressures, but as soon as possible waste filling is put in for permanent support.

The method is commonly employed for stoping upward from the level (fig. 13) but is sometimes used for mining laterally from a raise (fig. 14) or less often downward from a level (fig. 15). Where the walls and orebody itself are sufficiently strong, stull-sets are sometimes used in place of square-sets (fig. 16).

Application.- Square-set or stull-set stoping is applicable to the mining of irregular bodies of high-grade ore too weak to stand long without artificial support, even over small spans and with weak walls which require filling to support them and which can not be mined from the top downward due to the necessity of permanently supporting the back. It has an important application as an auxiliary to other methods as, for example, the mining out of pillars between filled stopes.

Advantages.-

1. Irregular ore bodies may be worked, leaving horses of waste or lean ore in place, and tongues and off-shoots of ore may be followed.

2. Waste may be sorted out in the stopes and ore dilution reduced to a minimum.

3. Control of grade of ore is easy, since each new face can be sampled and assayed before it is necessary to drill the face.

4. By filling the sets, only a small amount of ground need stand open at a time.

Disadvantages.-

1. High cost of mining per ton of ore.

2. Slow ore extraction and small tonnage mined per man-shift.

3. Large amount of timber required, cost of timber and labor of framing and placing it. Fire hazard wherever large quantities of timber are used.

4. Highest accident rate of any underground stoping method.

Results.- Two tables are given; one to show, for the different metals, the metal yield and man-hours consumed per ton mined by the square-set and stull-set stoping methods

at mines that produced over $100,000 in 1929; the other to give the details of labor, explosives, and power consumption and costs per ton mined at individual mines using this method.

Metal yield and man-hours consumed per ton mined by square-set and stull-set stoping methods at mines producing over $100,000 in 1929

Metal	Number of mines	Tons mined	Yield per ton	Man-hours per ton
Gold	7	379,022	0.35 oz.	5.35
Copper	26	4,318,294	92 lbs.	4.95
Iron	0	0	0	0
Lead-zinc	28	3,121,746	-	5.45

Details of labor, explosives, and power consumption and cost per ton at mines using square-set stoping methods

Stoping method	Mine	Ore	Width and dip of orebodies	Labor and supply consumption per ton: Labor, total underground, man-hrs	Explosives, pounds	Timber	Power, kw.h.	Direct mining costs per ton: Development	Stoping	Haulage and hoisting	General underground expense	Surface applicable to underground	Total	Refer I.C. list, p. 36
Square-set	Magma	Cu	20 to 30 ft.: 45 to 80°	4.260	0.900	19.0 bd. ft.	34.61	$0.736	$2.849	$0.616	$1.166	$0.130	$5.497	(55)
Square-set	U.V.X.	Cu	Wide bodies; steep	3.236	–	18.5 bd. ft.	–	.676	1.894	.508	.474	–	3.552	(56)
Square-set	Park-Utah	Pb, Zn, Ag	3 to 80 ft.; 40 to 55°	3.540	1.530	16.5 bd. ft.	29.23	.828	2.426	.282	.583	.543	4.662	(57)
Square-set	Argonaut	Au	Av. 20 ft., max. 65 ft.; 63°	4.168	1.010	7.05 bd. ft. 1.08 lin. ft.	33.08	.165	2.439	.519	.577	.291	3.991	(58)
Square-set	Tintic Standard	Pb, Ag	Wide bodies; 45 to 90°	7.255	1.068	25.5 bd. ft.	46.60	1.143	4.973	1.158	.322	.783	8.379	(59)
Square-set	Silver King	Pb, Zn, Au, Ag		8.055	3.607	(?)	77.70	2.086	3.502	1.848	.309	–	7.745	(61)
Square-set	Page	Pb, Zn, Ag	Max. 20 ft.; 40 to 60°	2.758	1.405	13.09 bd. ft.	15.07	1.170	2.150	.160	.180	.160	3.820	(62)
Square-set; cut-and-fill	Ground Hog	Pb, Zn, Cu	3 to 25 ft.; av. 50°	4.758	.630	9.00 bd. ft.	8.75	1.393	1.762	.628	.080	.127	3.990	(63)
Square-set	Black Rock	Pb, Zn, Cu, Au, Ag	Stopes 6 to 70 ft.; av. 80°	6.683	1.730	12.0 bd. ft. 2.8 lin. ft.	53.70	–	–	–	–	–	–	(60)
Square-set	Bunker Hill and Sullivan	Pb, Ag	Up to 40 ft.; 40 to 50°	3.705	$0.167 per ton	$0.321 per ton	$0.188 per ton	$11.43 per ft	2.361	distributed	distributed	distributed	3.897	(64.
Stull and post sets with fill; some shrinkage	Hecla and Star	Pb, Zn, Ag	3 to 40 ft.; 70 to 80°	2.640	0.783	11.207 bd. ft. 1.234 lin. ft	10.97	.638	1.211	.322	.766	.360	3.297	(70)
Stull and post sets with fill	Morning	Pb, Zn, Ag	6 to 30 ft.; 80 to 90°	3.129	.964	12.753 bd. ft.	39.37	.442	2.371	.530	.698	.204	4.245	(71)

Caved-Stope Methods

The caved-stope method is used where the overlying rock or capping of the orebody is too weak to form a roof that can be supported by pillars as in the open-stope method, and where permanent support of the roof or surface is not necessary. No attempt is made to support this capping, which is allowed to cave as the extraction of the ore proceeds directly beneath it. Under caved stopes, we have three principal subdivisions; two of them are distinctly different methods of stoping and the other is a combination of these two.

First, we have top slicing (77), in which the ore is broken by drilling and blasting, and after its removal from the stope the capping or overburden is caved. In this method, mining starts at the top of the ore body and advances progressively downward, each successive slice being mined out right up to the cave above it.

Second, we have the block-caving method (103) in which the ore is undercut at a considerable distance below the capping and is broken up by caving action induced by the force of gravity, both broken ore and capping moving downward in a mass as the broken ore is drawn off from below.

The third method is termed "sublevel caving" (77). In this method mining starts near the top of the ore body, and slices are taken similar to those driven in top slicing; however, instead of spacing the levels so that each slice is mined up to the cave above, the vertical interval is made greater, the slice being driven at ordinary drift height to the stope limit, and the back of ore over the slice and under the cave is broken down on the retreat by caving, assisted to some degree by light blasts. This method thus resembles top slicing in two respects and block caving in another, but it can not, strictly speaking, be classed with either one.

Top slicing probably originated in the iron mines in the north of England and was first applied in this country, as far as is known, at the Lake Angeline mine, Ishpeming, Mich., some 50 years ago, by Capt. Thomas Walters. It has found its widest application in this country in the iron mines of the Lake Superior region and is the standard method of underground mining on the Mesabi range.

Sublevel caving was probably developed from top slicing and is the type method employed on the Gogebic range, in Michigan.

Block caving was early employed, at least as far back as the nineties, at the Pewabic mine on the Menominee range in Michigan, and later at the Tobin mine. It was later applied in Utah and subsequently at the porphyry copper deposits of Arizona (100 to 104) and Nevada, where it has reached its highest efficiency and found its greatest use. The method is also employed in Chile (105) at the Andes Copper Co.'s mine and is to be applied at Mt. Isa in Queensland.

<u>Results</u>.- Four tables are given; one shows, for the different metallic ores, the metal yield and man-hours consumed per ton mined by caved-stope methods at mines that produced over $100,000 in 1929; the others give the details of labor, explosives, and power consumption and costs per ton mined at individual mines using sublevel-caving, top-slicing, and block-caving methods.

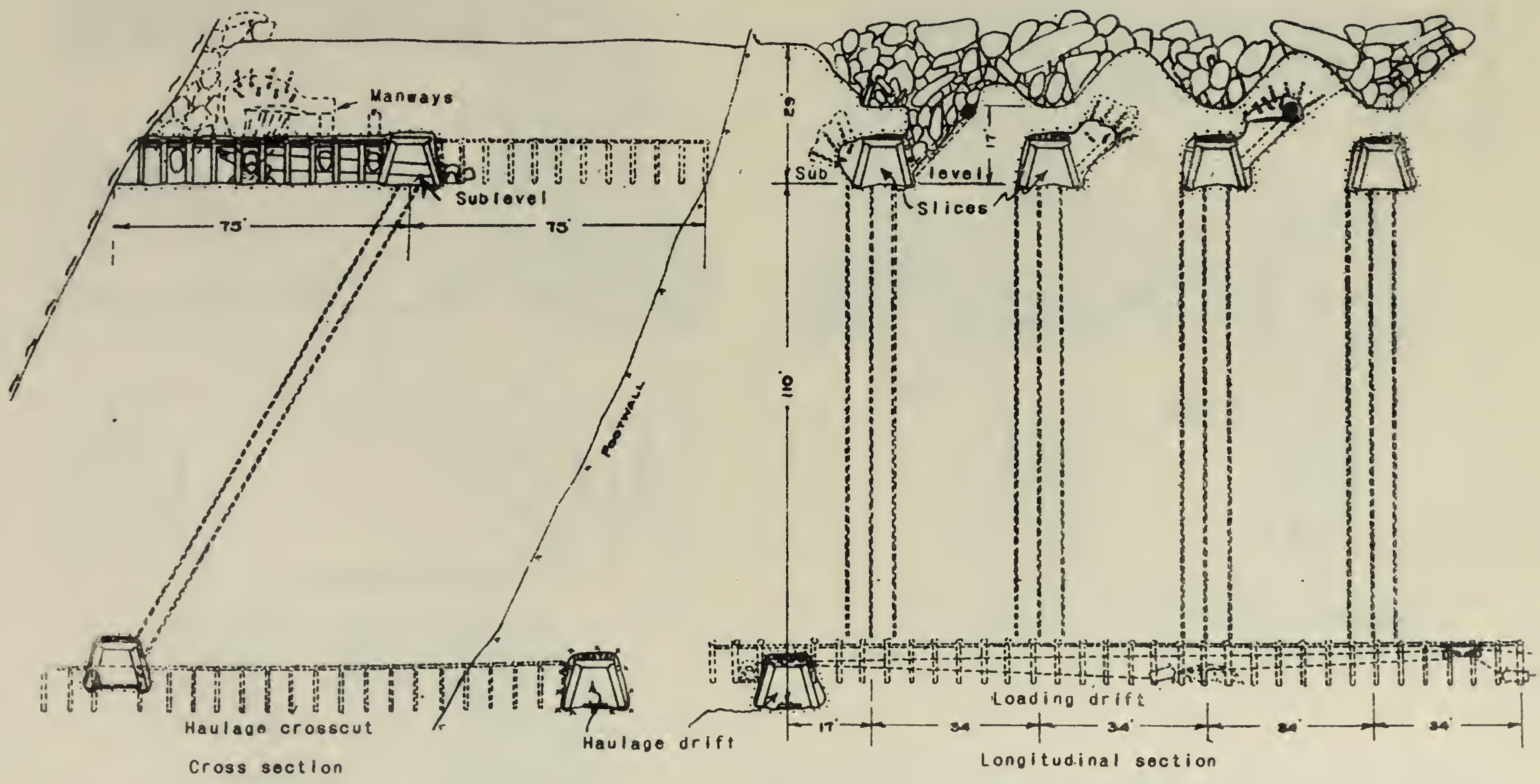
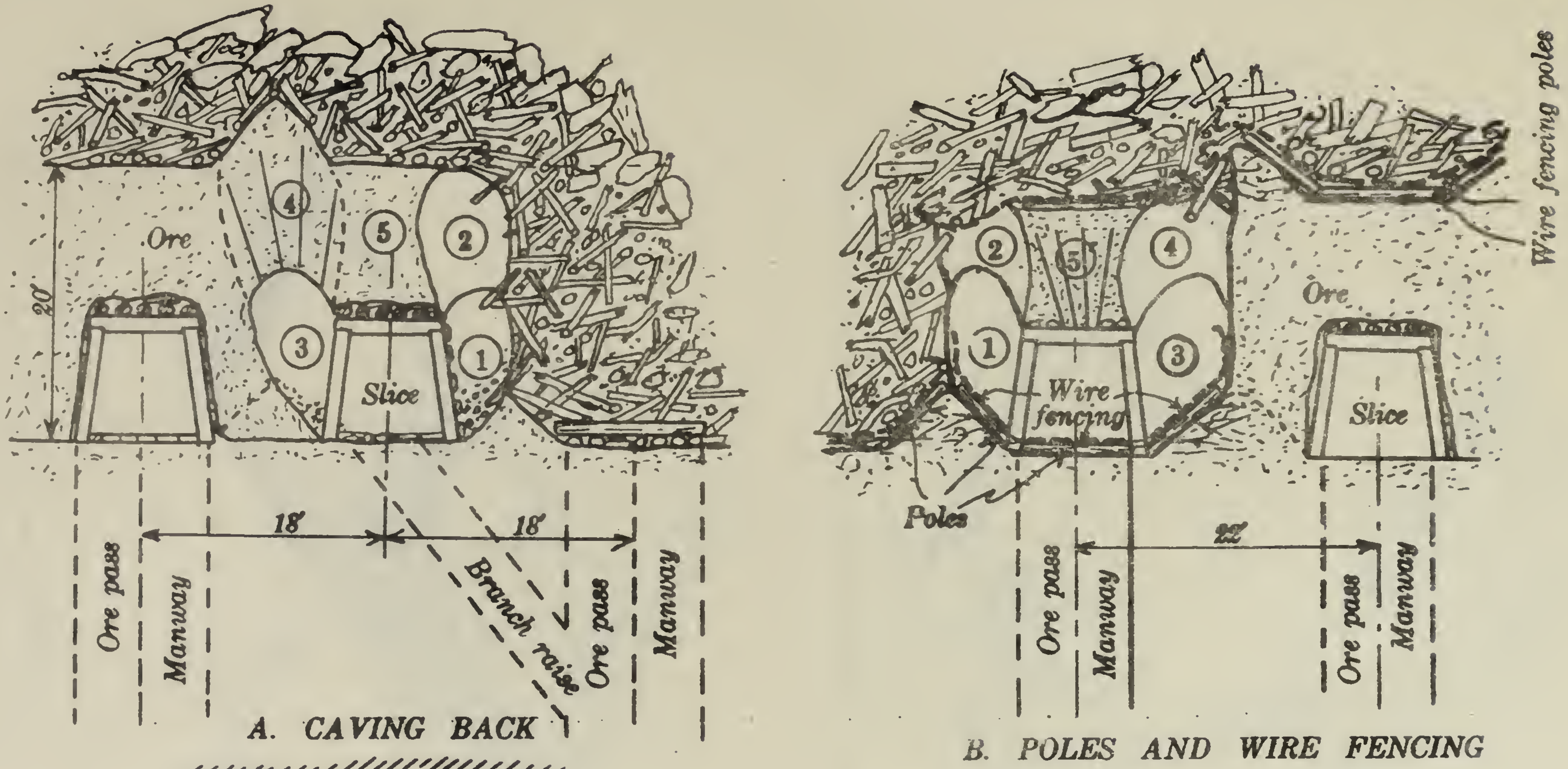


Figure 17.- Sublevel caving with scraper loading drift



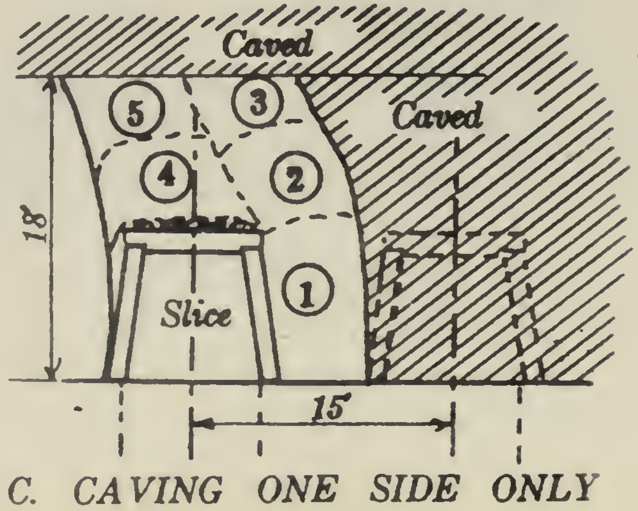


Figure 18 -- Cross sections showing "caving back" operation

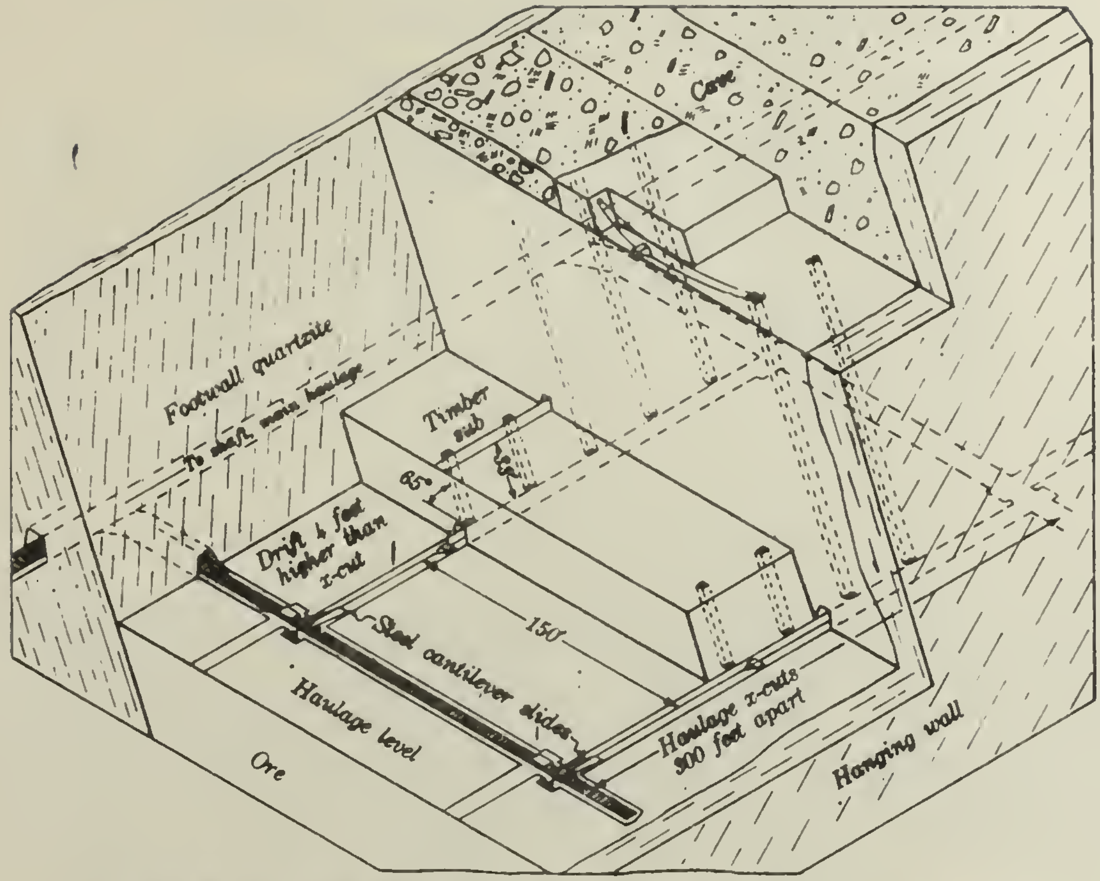


Figure 19.- Sublevel caving method of mining using scrapers

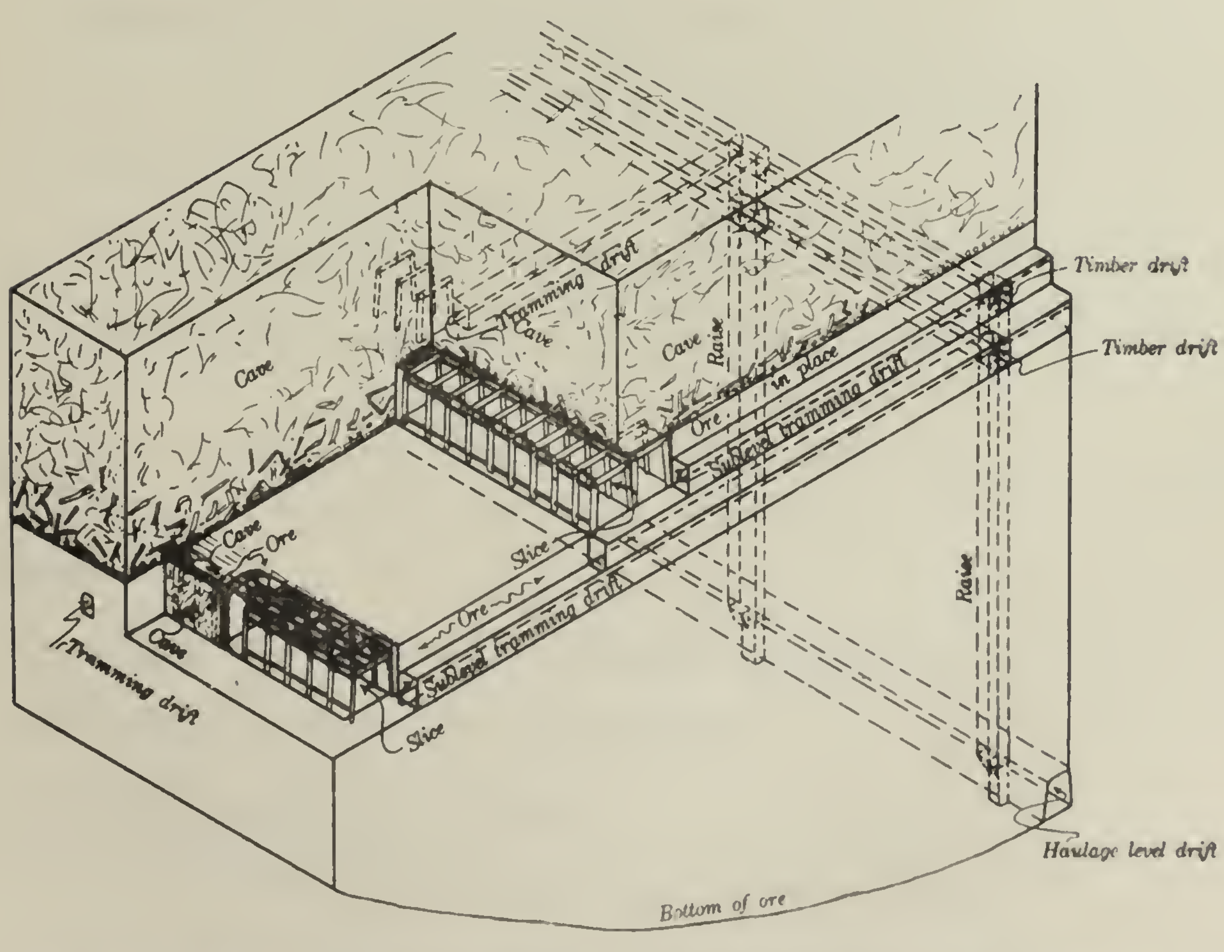


Figure 20.- General scheme of horizontal top slicing using sublevel tramming drifts

Metal yield and man-hours consumed per ton mined by caved-stope methods at mines producing over $100,000 in 1929

Metal	Number of mines	Tons mined	Yield per ton	Man-hours per ton
Gold	0	0	0	0
Copper[3]	7	18,576 196	18.6 lbs.	0.63
Lead-zinc	0	0	0	0
Iron[1]	37	14,200,257	.520 ton	1.05
Iron[2]	17	6,521,407	.529 ton	1.06

1/ Top slicing; 2/ sublevel caving; 3/ block caving.

Sublevel Caving (77, 90, 17)

Application.- Sublevel caving is applicable to the mining of low to medium grade steeply dipping bodies of fairly weak and moderately soft ore inclosed in walls of weak to medium strength and under a moderately weak but hard capping which will cave in large blocks. Ore bodies may be wide or narrow. The conditions required are similar to those for top slicing except that for top-slicing the capping must break fairly small and cave immediately the support is removed and fill the excavation tightly, whereas for sublevel caving the capping should hang for a short time over small spans while the caved ore is being removed (figs. 17, 18, 19). Obviously it must be permissible to cave the surface.

Advantages.-

1. Low cost per ton for mining heavy ground.

2. Lower timber cost and less stope development per ton than for top slicing.

3. Rapid ore extraction.

4. Under certain conditions, high percentage of extraction and very littl dilution.

5. Applicable to ores of more or less wet and sticky nature not suited to block caving.

6. Lowest accident rate (United States mines, 1929) of any of the underground stoping methods.

Disadvantages.-

1. More dilution than with top slicing and square-setting.

2. Practically no sorting in stopes possible.

3. Ventilation not easily provided.

4. Some fire hazard due to accumulation of timber in the gob.

5. Low grade or "Junore" in capping and marginal areas is lost.

Details of labor, explosives, and power consumption and costs per ton mined at mines using sublevel-caving methods

				Labor and supply consumption per ton				Direct mining costs per ton						
Stoping method	Mine	Ore	Width and dip of orebodies	Labor, total under-ground, man-hrs	Explo-sives, pound	Timber	Power, kw.h.	Devel-opment	Stoping	Haulage and hoist-ing	General under-ground expense	Surface applic-able to under-ground	Total	Refer I.C. list, p. 36
Sublevel caving	Montreal	Fe	Scattered bodies, narrow to wide; 65°	1.336	0.590	2.76 bd. ft.	6.24 plus 32 lbs. coal	$0.254	$0.360	$0.344	$0.311	$0.070	$1.399	(17)
Sublevel caving	Eureka	Fe	5 to 20 ft. up to wide bodies; 55 to 75°	1.397	.710	3.35 bd. ft.	15.62	.317	.398	.408	.414	-	1.537	(90)
Sublevel caving	No. 16	Fe	10 to 175 ft.; 65°	1.193	.421	1.55 bd. ft. 0.0013 cords lagging	-	-	-	-	-	-	-	-
Sublevel caving	No. 20	Fe	110 ft.; 65°	.971	.697	(?)	9.38	-	-	-	-	-	-	-

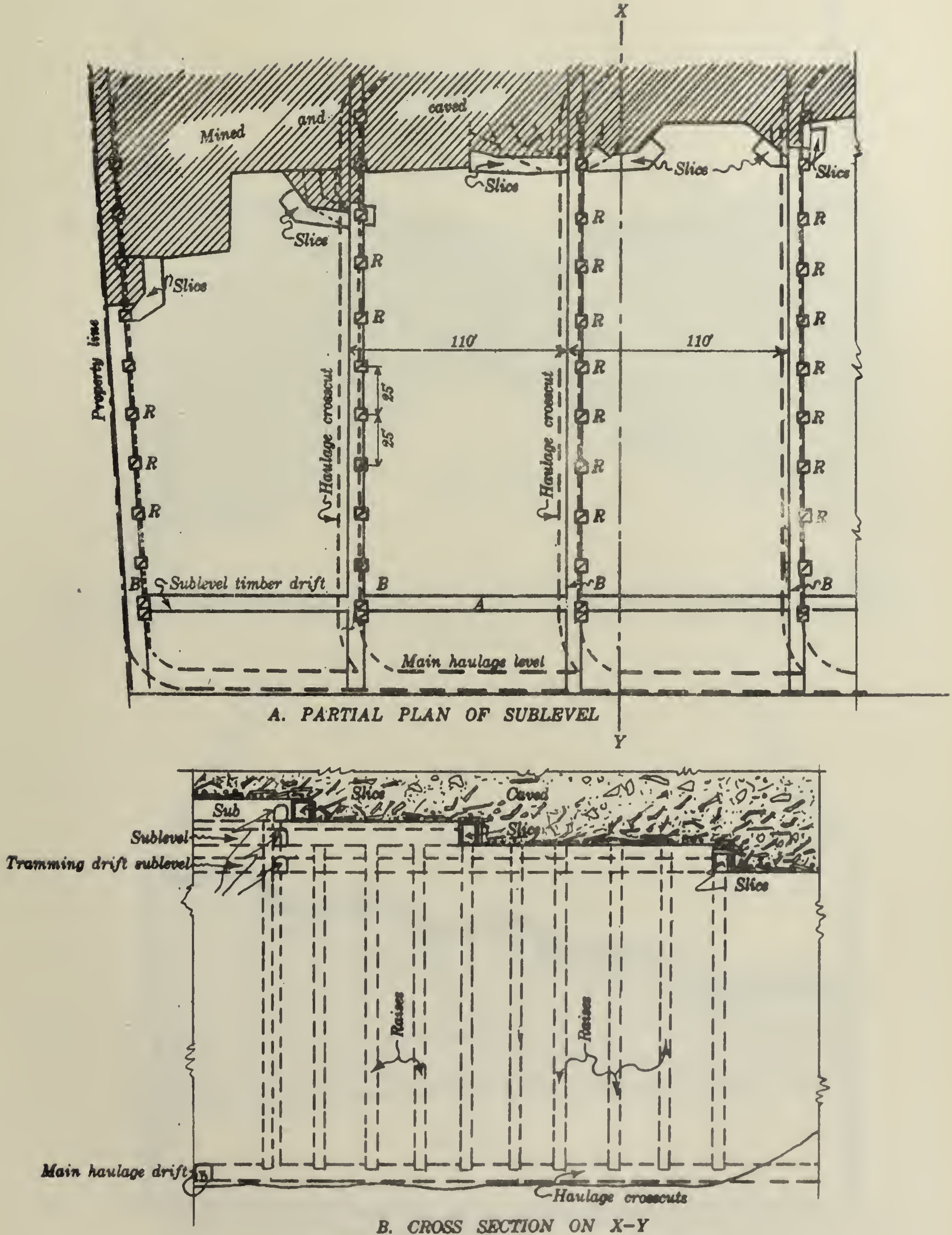
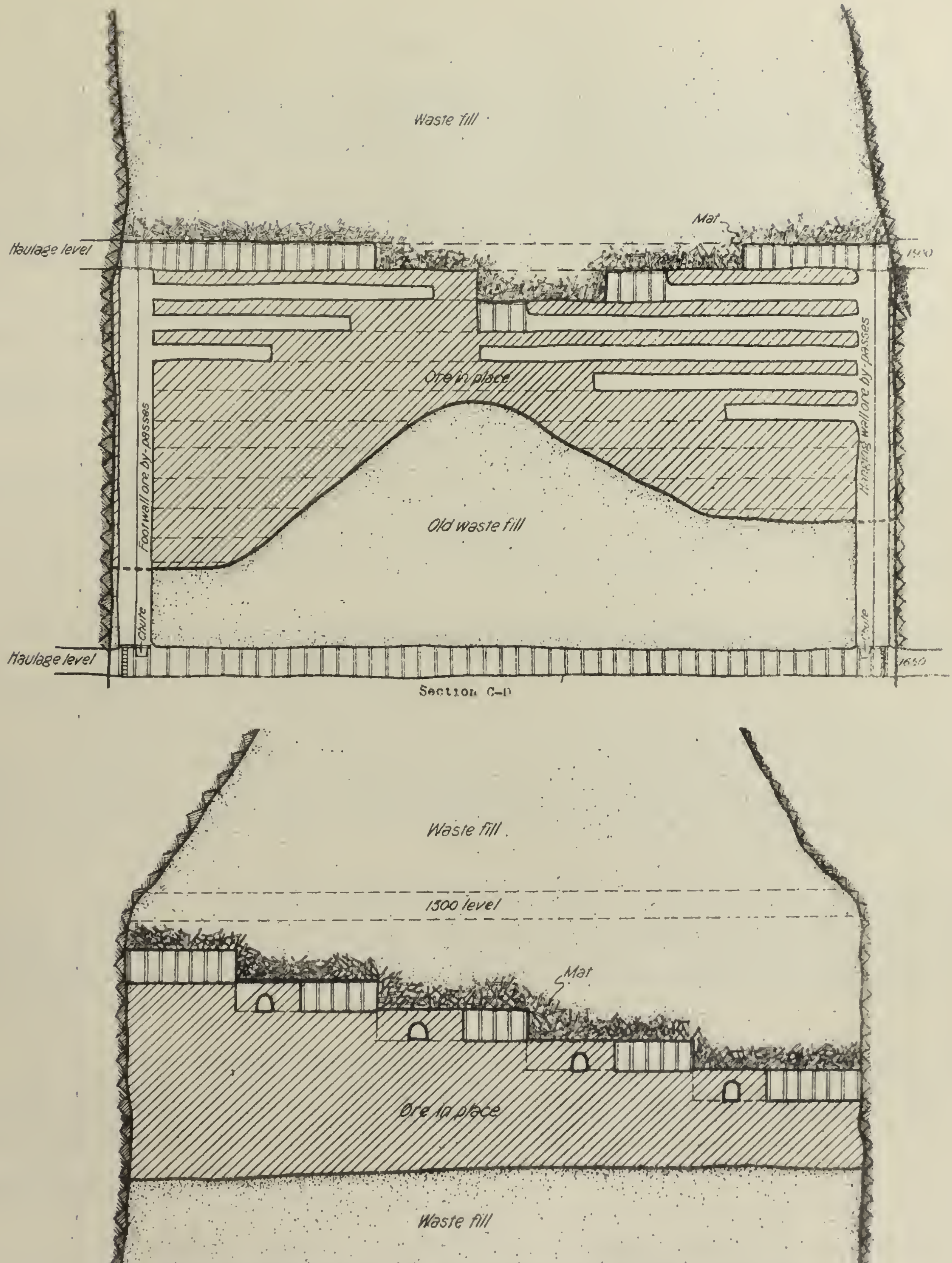


Figure 21 – Radial top slicing to chutes; thick ore body



Section A-B

Figure 22.- Vertical sections through No. 2 top slice stope United Verde Mine. Arizona

Top Slicing (75 to 77)

Application.- Top slicing is applicable to wide or fairly wide orebodies too weak to stand without support except over small spans and having a weak capping which will cave readily when the support is removed from beneath it and fill the excavated space tightly. The side walls may be strong or weak if the ore body is wide, but for narrow orebodies the hanging wall should be weak. Suitable timber should be available in large quantities at low or moderate cost. (Figs. 20, 21, 22.) It is obvious that caving and subsidence of the surface must be permissible.

Top slicing also has an important application in the mining out of pillars and for robbing operations in caved or broken ground.

Advantages.-

1. Top slicing is a safe method of mining heavy ground which can not be economically and safely worked by overhand-stoping methods, and is economical where timber is plentiful and reasonable in cost.

2. A very high extraction of the ore is possible with practically no dilution from the capping and walls.

3. Under proper supervision the method is safe, as evidenced by the many excellent safety records made by mines employing it. The accident rate is the second lowest for underground methods in the United States.

4. Where market conditions are such that the mine is faced with occasional shutdowns, the stopes or slices may be blasted down so that the ore body remains in good condition for resumption of operations on short notice even after idleness of considerable duration, although development headings may require considerable repair work and maintenance if close to the caved ground.

5. Although considerable preparatory development is required prior to beginning slicing operations, once this development is well started a considerable tonnage of ore is produced therefrom, and in the type of ore to which top slicing is commonly applied the cost of development ore is not greatly in excess of that of ore from slicing (see items 6 and 7, under Disadvantages).

6. Top slicing can be employed under sand and other loose surface material and does not require as clean a mat as does sublevel caving. If fine waste is mixed with the gob in sublevel caving the ore is apt to be considerably contaminated thereby (see item 10, under Disadvantages).

Disadvantages.-

1. If timber and lagging are not available in adequate supply and are expensive, the top-slicing method has obvious disadvantages from the standpoint of cost.

2. The method is more expensive than the block caving or shrinkage which may be used under similar conditions, though with these methods a lower ore extraction and higher dilution of the ore are apt to result.

3. Where it is necessary to protect the surface against subsidence, this method would seldom be applicable, although under certain deep-mining conditions where the ore body is thin it might still be employed. Sometimes fault planes, jointing, or other lines of weakness in the capping may cause subsidence at some distance from the area immediately over the mine, while in other cases only the area directly over the excavation may be affected.

4. Ventilation is not simple with top slicing in deep mines where work is being done under a thick mat of timber; in heavy sulphide ores which oxidize when exposed to the mine air with attendant generation of heat and possibly of gas, ventilation may be an important consideration. In some cases the only solution is to serve each slice with a small, individual, booster fan, as recirculation of vitiated air must be guarded against. Forced ventilation from the surface with fans of large capacity is used in some of the deeper mines.

5. Where large amounts of old timber accumulate, as in top-slice operations, a certain fire hazard exists. Fortunately many mines employing top slicing are wet or decidedly moist, and gob fires are not of common occurrence.

6. In order to obtain a large output, a considerable number of working places are required. This may mean that working places are scattered over a considerable area and connected by a correspondingly large amount of development openings.

7. The period of development prior to production from slicing is relatively long. Rate of output can not be suddenly increased to meet market demands unless development is considerably in advance of normal production requirements.

8. Timbering and covering down the slices consumes a large part of the shift, which reduces the available time for breaking and mucking. Thus the drills and scraping equipment are idle much of the time.

9. Handling of timber, lagging, boards, and wire fencing is a large item of expense in top-slice mining, since these materials are used in large quantities and are often transported long distances in the mine.

10. If the capping breaks in large blocks which wedge together so as to leave a large open space below them, the possibility of sudden collapse presents a serious element of danger to the slicing operations below, especially if there is not a good cushion of gob between the open space and the top of the active slice (see item 6 under Advantages).

Details of labor, explosives, and power consumption and costs per ton mined at mines using top-slicing methods

Stoping method	Mine	Ore	Width and dip of orebodies	Labor and supply consumption per ton: Labor, total underground man-hrs	Explosives, pound	Timber	Power, kw.h.	Direct mining costs per ton: Development	Stoping	Haulage and hoisting	General underground expense	Surface applicable to underground	Total	Refer I.C. list, p. 36
Top slicing and sublevel stopes	Marquette No. 2	Iron	50 to 100 ft.; 40 to 90°	1.029	0.473	0.55 lin. ft.	9.39	$0.251	$0.707	$0.149	$0.193	$0.066	$1.366	(11)
Top slicing	Mesabi No. 1	Iron	Wide, but only 10 to 30 ft. thick	1.174	.395	2.83 bd. ft.	4.74	(.729	)	.128	.250	.058	1.165	(75)
Top slicing	Marquette No. 5	Iron	100 ft. thick, 400-800 ft. wide	.903	.408	.672 lin. ft.	12.49	.052	.739	.210	.090	.004	1.095	(76)
Top slicing	Mesabi No. 2	Iron	Av. 15 ft. thick	1.154	.801	(?)	2.97	-	-	-	-	-	-	-
Top slicing	Mesabi No. 3	Iron	Av. 15 ft. thick; 3½°	.731	.694	1.24 lin. ft. 0.0012 cord lagging	1.86	-	-	-	-	-	-	-
Top slicing	Mesabi No. 4	Iron	Av. 40 ft. thick; 4°	.777	.463	1.06 bd. ft. 1.26 lin. ft.	3.49	-	-	-	-	-	-	-
Top slicing	Mesabi No. 5	Iron	Av. 40 ft. thick; 5°	.993	.553	1.226 bd. ft. 0.00164 cord lagging	3.36	-	-	-	-	-	-	-
Top slicing	Mesabi No. 6	Iron	1000 ft.; 6°	1.127	.562	2.21 lin. ft. 0.00325 cord lagging	4.16	-	-	-	-	-	-	-
Top slicing	Mesabi No. 7	Iron	300 ft.; 60 to 80°	1.263	.433	(?)	5.25	-	-	-	-	-	-	-
Top slicing	Mesabi No. 8	Iron	10 to 75 ft.; steep	.972	.634	1.746 lin. ft.	11.51	-	-	-	-	-	-	-
Top slicing	Mesabi No.11	Iron	Wide ore; 45°	1.386	.547	1.702 bd. ft. 3.94 lin.ft. lagging	9.39	-	-	-	-	-	-	-
Top slicing	Mesabi No 12	Iron	Wide ore; 45°	1.048	.437	1.12 bd. ft. 1.85 lin. ft. poles; 4.47 lin.ft. lagging	18.41	-	-	-	-	-	-	-
Top slicing	Mesabi No.13	Iron	Wide ore; 45°	1.018	.365	1.376 bd. ft. 3.05 lin.ft. lagging	(?)	-	-	-	-	-	-	-

Block Caving (100 to 104)

Application.- Block caving is applicable to the mining of large thick bodies of low-grade ore of such physical character that, in caving and drawing, the ore will break up fine enough to pass the mill holes and will not pack in the chutes, and where the ratio of overburden to ore volume is too high to make opencut mining economical. It is obvious that caving and subsidence of the capping and surface must be allowable (figs. 23, 24, 25, 26, 27).

Advantages.-

1. Low mining cost per ton, approaching that for open-pit mining.

2. High rate of output is possible once the mine has been developed to the extraction stage.

3. Possibility of standardizing various operations to a high degree, thus making for efficiency and safety in mining.

4. Accident rate fairly low.

Disadvantages.-

1. Large capital expense and long period of time required for development and bringing mine into production stage.

2. Dilution of ore with waste and loss of ore, particularly around margins of an irregular orebody.

3. Necessity for very close supervision and extreme care in control of drawing operations.

4. Low grade of "Junore" in capping or marginal areas is lost.

In block caving, three things must be constantly watched and controlled:

First, the mining must be done in such a manner that excessive weight does not develop on the grizzly level. This is generally controlled by the rate at which the ore is pulled. Also, weight may develop on a line of grizzlies ahead of the undercutting. The reason for this is not definitely understood, although it can be largely prevented in most cases.

Second, the ore should be broken sufficiently in caving to be subsequently handled without further breaking. The slower the drawing rate, generally, the more the ore is broken up; also the slower the drawing rate, the more weight develops on the grizzly level. Generally a balance is taken between the two to get maximum efficiency. Should the undercutting not be done in the proper manner, a large block of ore may settle down on the undercutting level without being broken up. When this happens the expense of mining is greatly increased, or part of the orebody may be lost.

Third, the ore must be drawn in such a manner as to get maximum recovery of ore and minimum dilution with waste. The dilution may be both of the capping and of marginal rock.

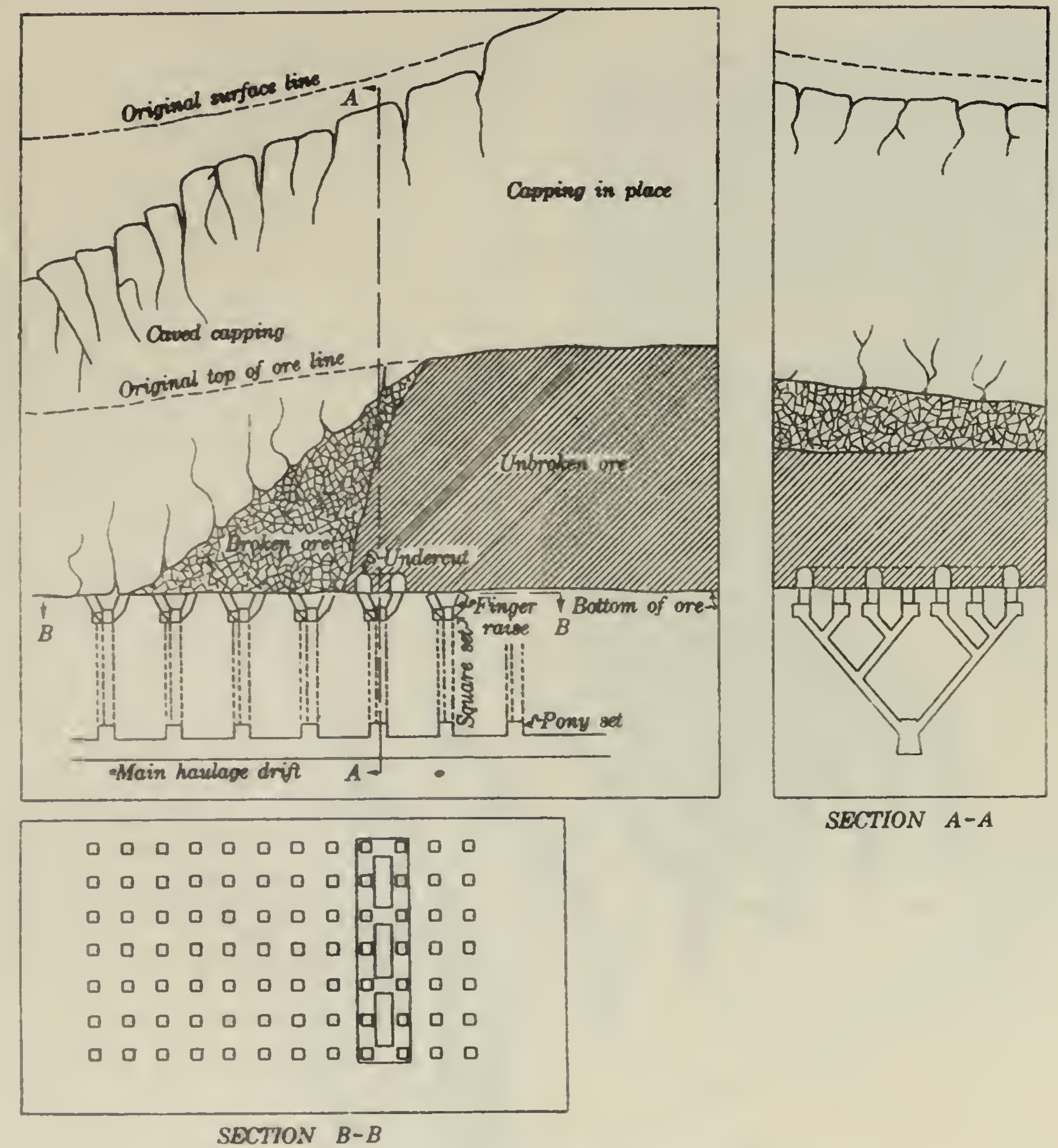


Figure 23 – Caving system used at Ruth mine Nevada

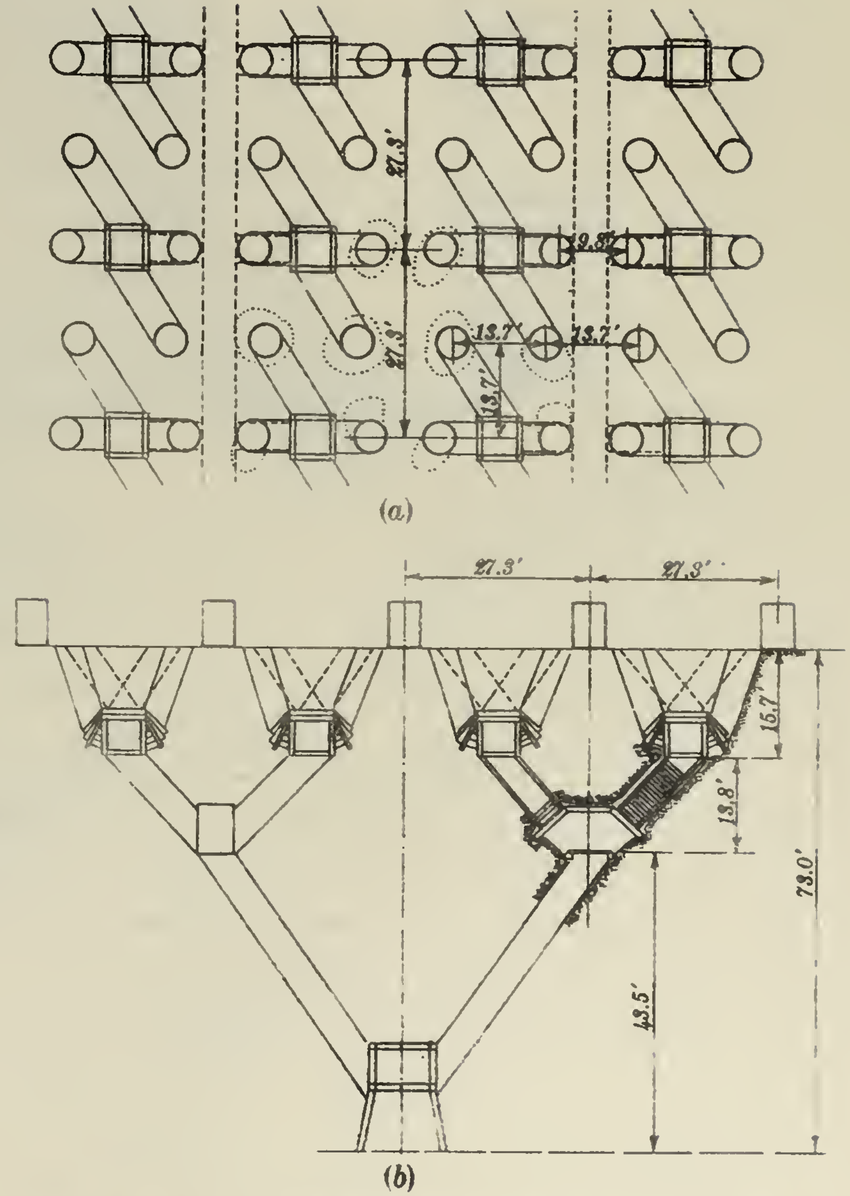


Figure 24 – (a) Plan of undercutting level showing top of branch raises. (b) Transverse section on main and branch raise system. Andes Copper Co.

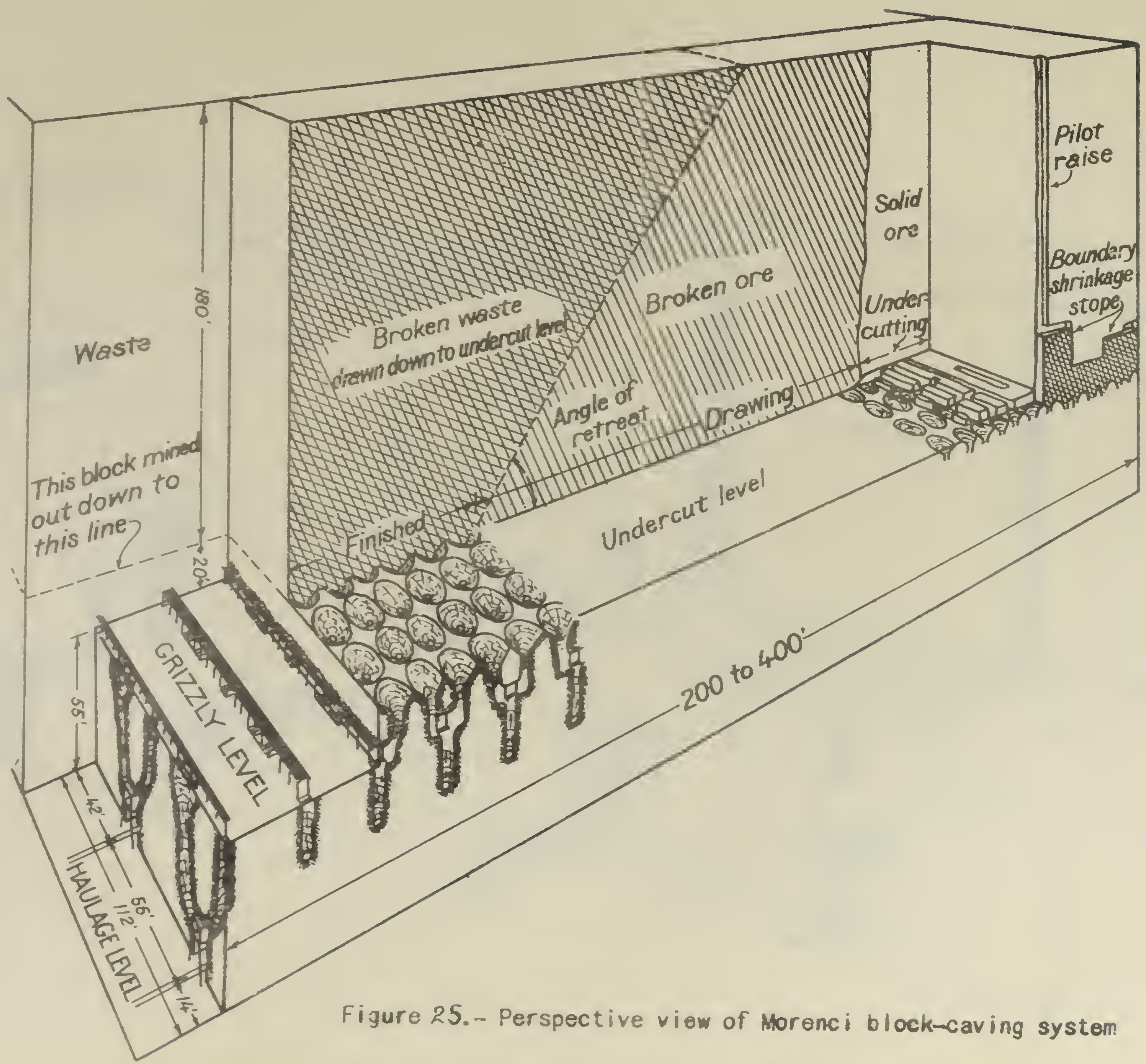


Figure 25.- Perspective view of Morenci block-caving system

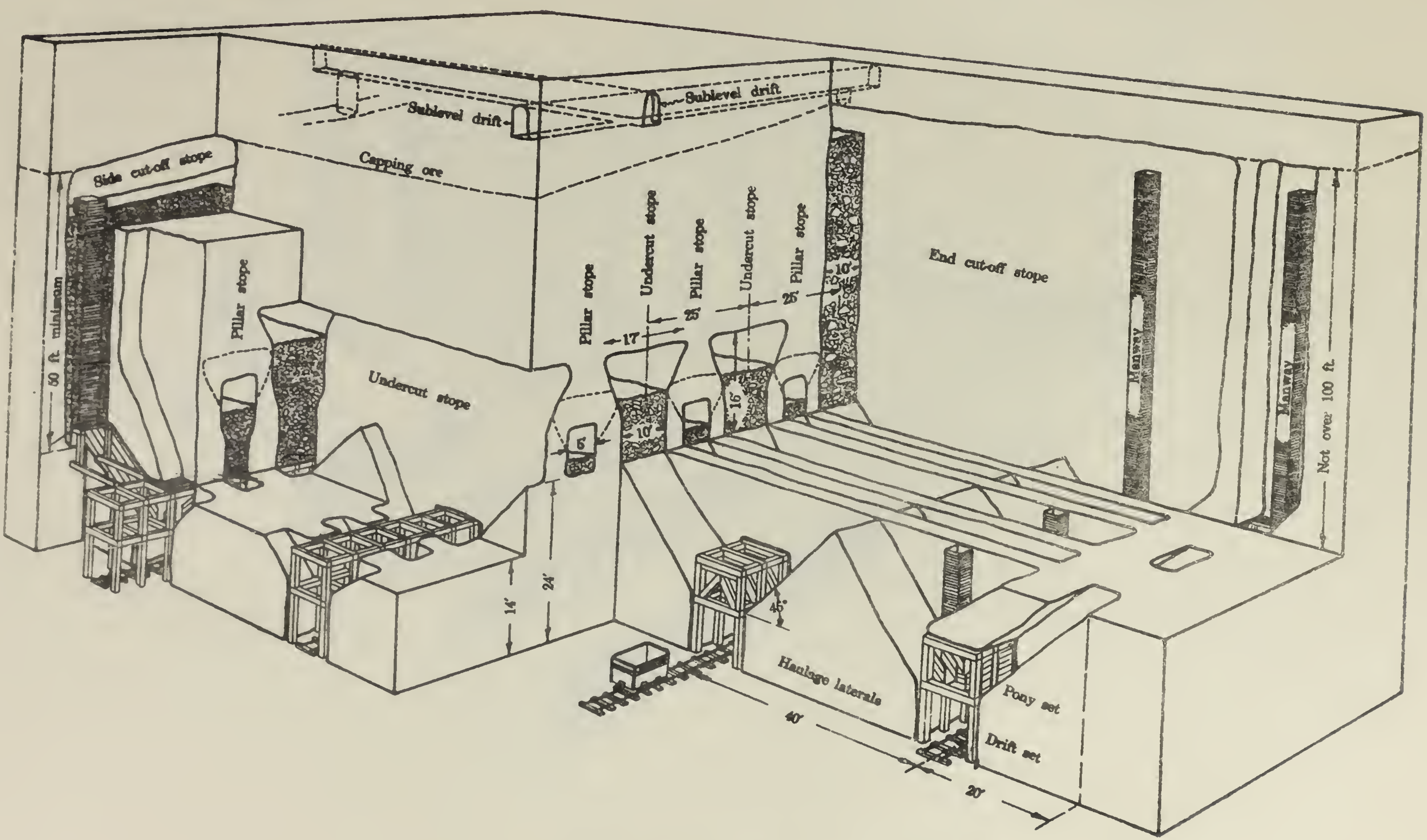


Figure 26 - Motor-haulage caving method

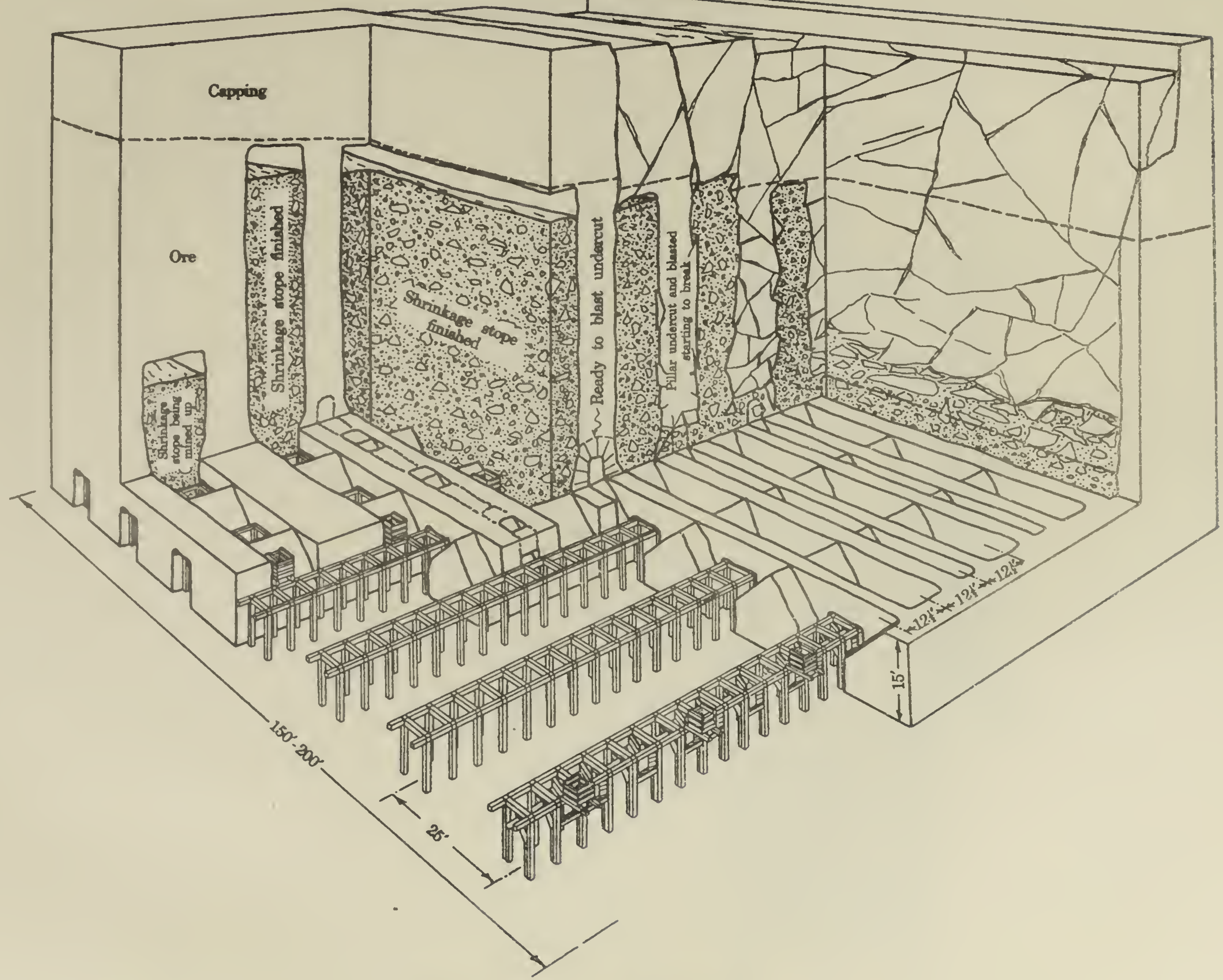


Figure 27. – Hand-tramming caving method

Details of labor, explosives, and power consumption and costs per ton mined at mines using block-caving methods

Stoping method	Mine	Ore	Width and dip of orebodies	Labor and supply consumption per ton: Labor, Total under-ground, man-hrs	Explo-sives, pound	Timber	Power, kw.h.	Direct mining costs per ton: Devel-opment	Stoping	Haulage and hoist-ing	General under-ground expense	Surface applic-able to under-ground	Total	Refer I.C. list, p 36
Block caving	Morenci	Cu	Large, massive	0.823	0.190	0.25 bd. ft.	-	$0.132	$0.108 est.	$0.100	($0.116)		$0.456	(100)
Block caving	Ray	Cu	Large, massive	.729	.319	1.88 bd. ft.	.104	.243	.111	.166	-		.624	(101)
Block caving	Inspiration	Cu	Large, massive	.625	-	-	-	.211	.208	.144	.012	$0.008	.583	(102)
Block caving	Miami	Cu	Large, massive	.384	.222	1.045 bd. ft.	1.90	.100	.136	.086	.057	.020	.399	(101)

Combined Underground Methods

Considerable objection has been offered to the inclusion of "combined" methods in a classification of stoping methods. However, where two or more of the typical methods are employed together in a regular scheme for mining each block of ore in a mine, it is the author's opinion that such combinations are deserving of sufficient prominence to be classified as distinct methods. This is especially true where the method contributing the greater part of the tonnage is dependent upon the other method for its successful application. Thus we have block caving combined with shrinkage stoping (fig. 28) in which the individual blocks to be caved are isolated one from the other by marginal shrinkage stopes.

Figure 29 shows shrinkage stoping combined with square-setting where, in the regular scheme of things, stopes are mined by shrinkage and each stope is separated from its neighbors by a pillar, the pillars being mined later by square-setting. Shrinkage stopes combined with pillar caving, in which stopes are separated by pillars that are later caved and drawn with the emptying of the stopes, are shown in Figure 30. We also have stopes separated by pillars; the stopes are mined by shrinkage with subsequent waste filling, and the pillars are later mined by top slicing (fig. 31). In each of these instances the major method is dependent upon the auxiliary method for its successful application.

Opencut Methods (110 to 115)

Opencut methods of mining have a wide use in the mining of surficial deposits or wide deposits having a relatively thin capping (figs. 32, 33, 34, 35). Broadly speaking, they are applicable where the ratio of overburden to ore and the nature of the overburden are such that the total cost of removing the overburden plus the cost of mining the ore by open-pit methods is less than that of mining by underground methods, such items as capital investment and interest on development expense during removal of overburden prior to the productive life of the property being taken into consideration.

Iron ore, copper ore, coal, phosphate rock, stone, sand and gravel, and other nonmetallics are mined in large quantities by opencut methods. Placer mining (except in drift placers) is also a form of opencut mining.

Opencut operations include those using hand methods, steam and electric power shovels (110, 111, 112), and drag lines. Even a brief description of each of these methods, having many variations as they do, would be too voluminous for the present paper.

Advantages.-

Opencut methods, where they can be applied, have the obvious advantage of being conducted in daylight, and where a night shift is employed flood lights may be used.

1. Rates of output may be varied within a wide range without corresponding variations in operating costs.

2. Low cost of operation on the average.

3. Mechanical methods of loading and haulage are easily applied, resulting in small labor requirements per ton produced, and ease of increasing production without large additions to operating forces.

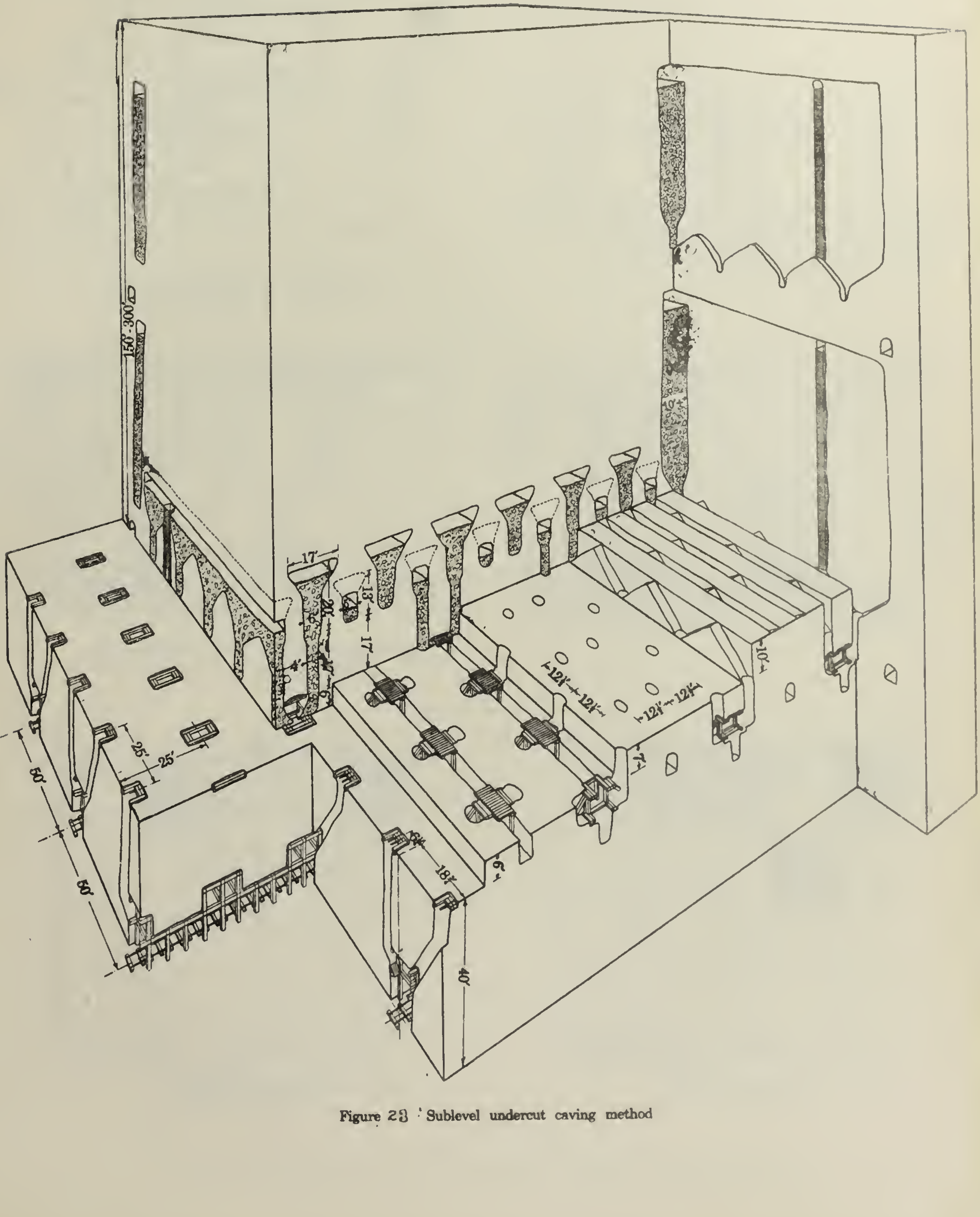
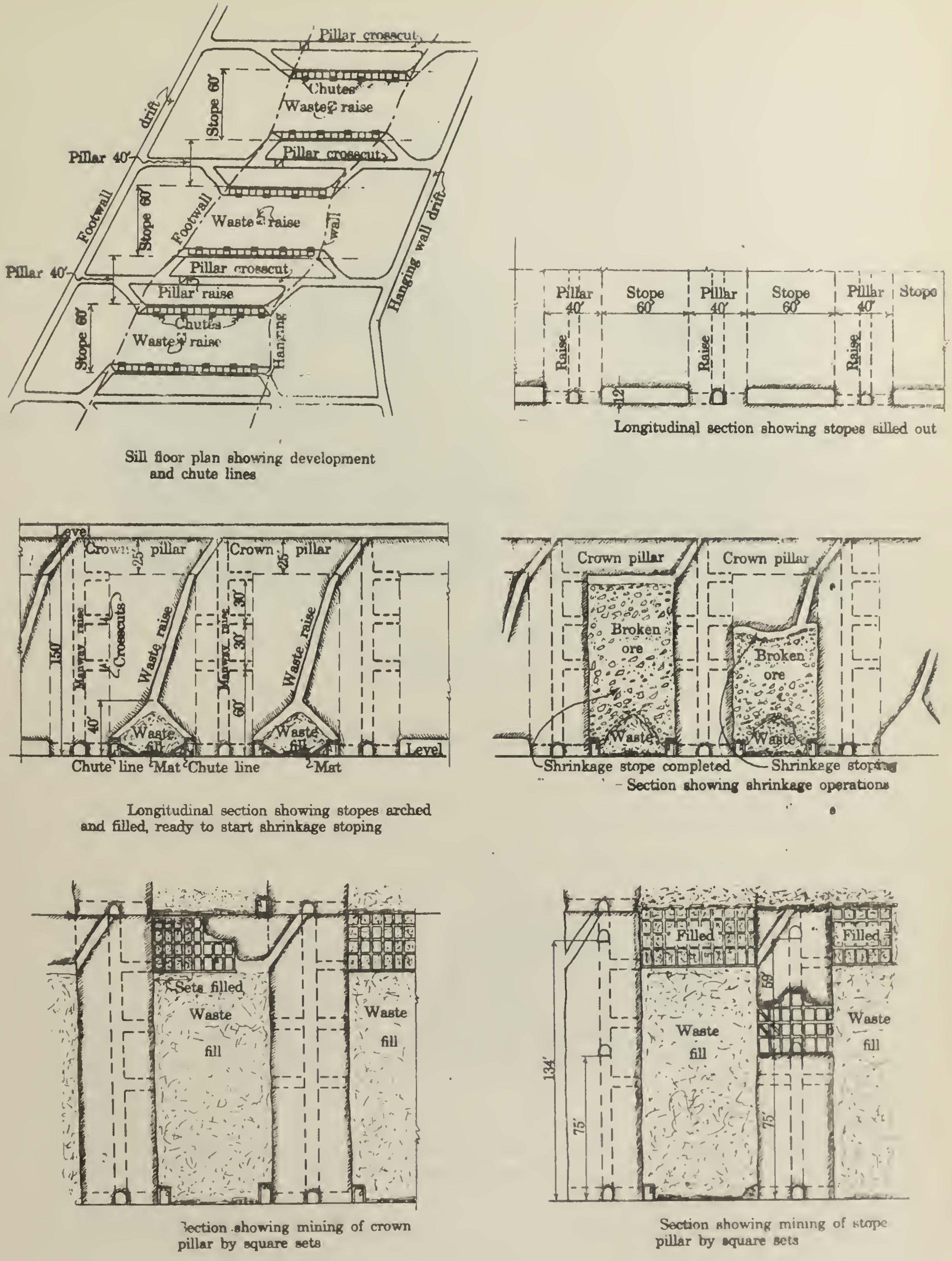


Figure 23 · Sublevel undercut caving method



Sill floor plan showing development and chute lines

Longitudinal section showing stopes silled out

Longitudinal section showing stopes arched and filled, ready to start shrinkage stoping

Section showing shrinkage operations

Section showing mining of crown pillar by square sets

Section showing mining of stope pillar by square sets

Figure 29.- Combined methods; shrinkage stoping combined with square-setting.

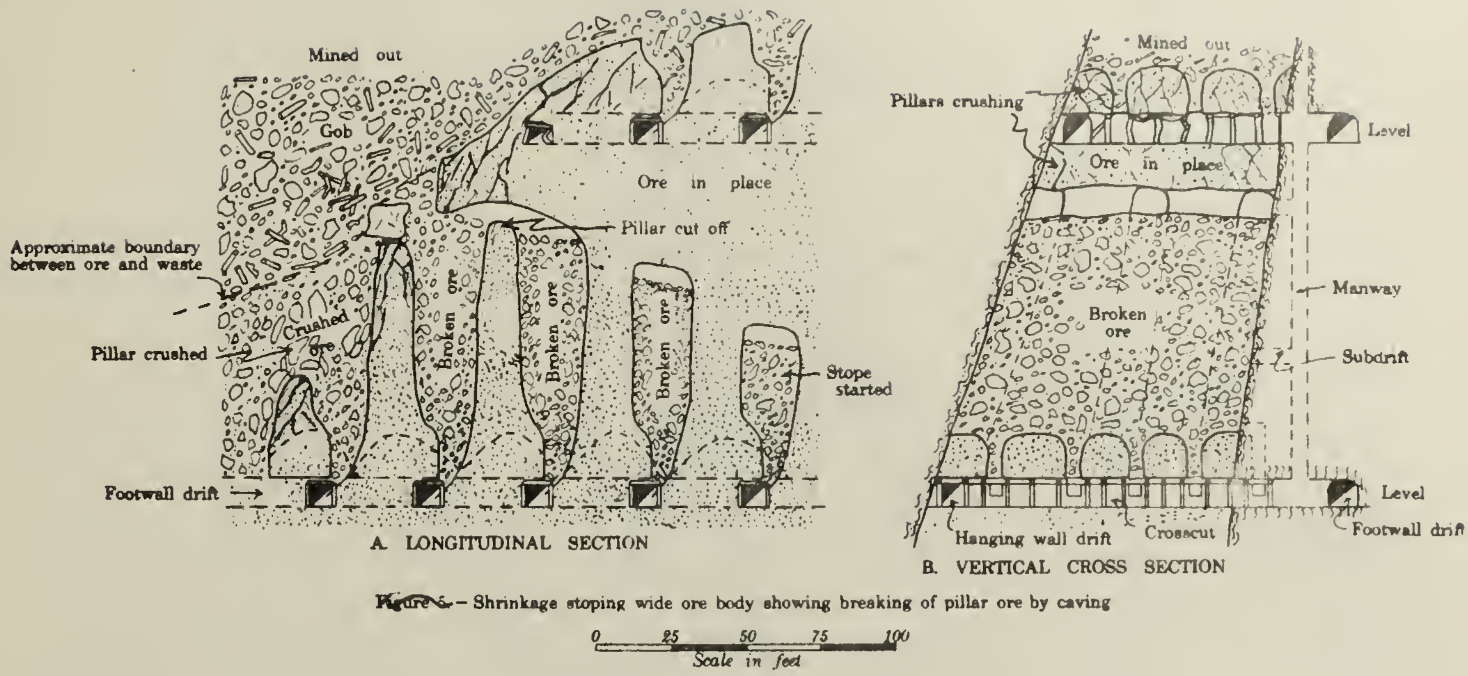


Figure 30.- Combined methods; shrinkage stoping combined with pillar caving

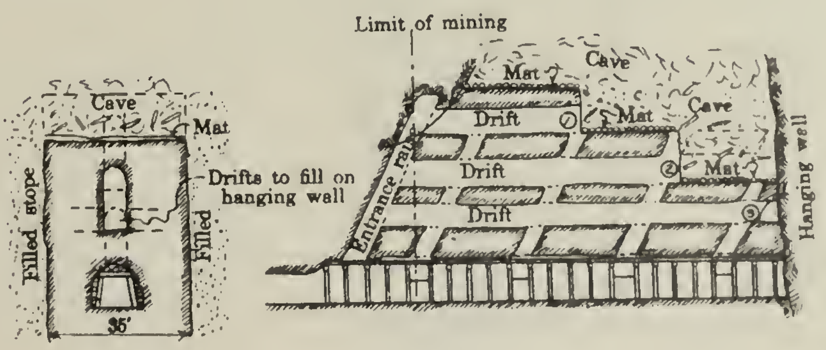


Figure 31.- Combined methods; cut-and-fill stoping combined with top slicing of pillar

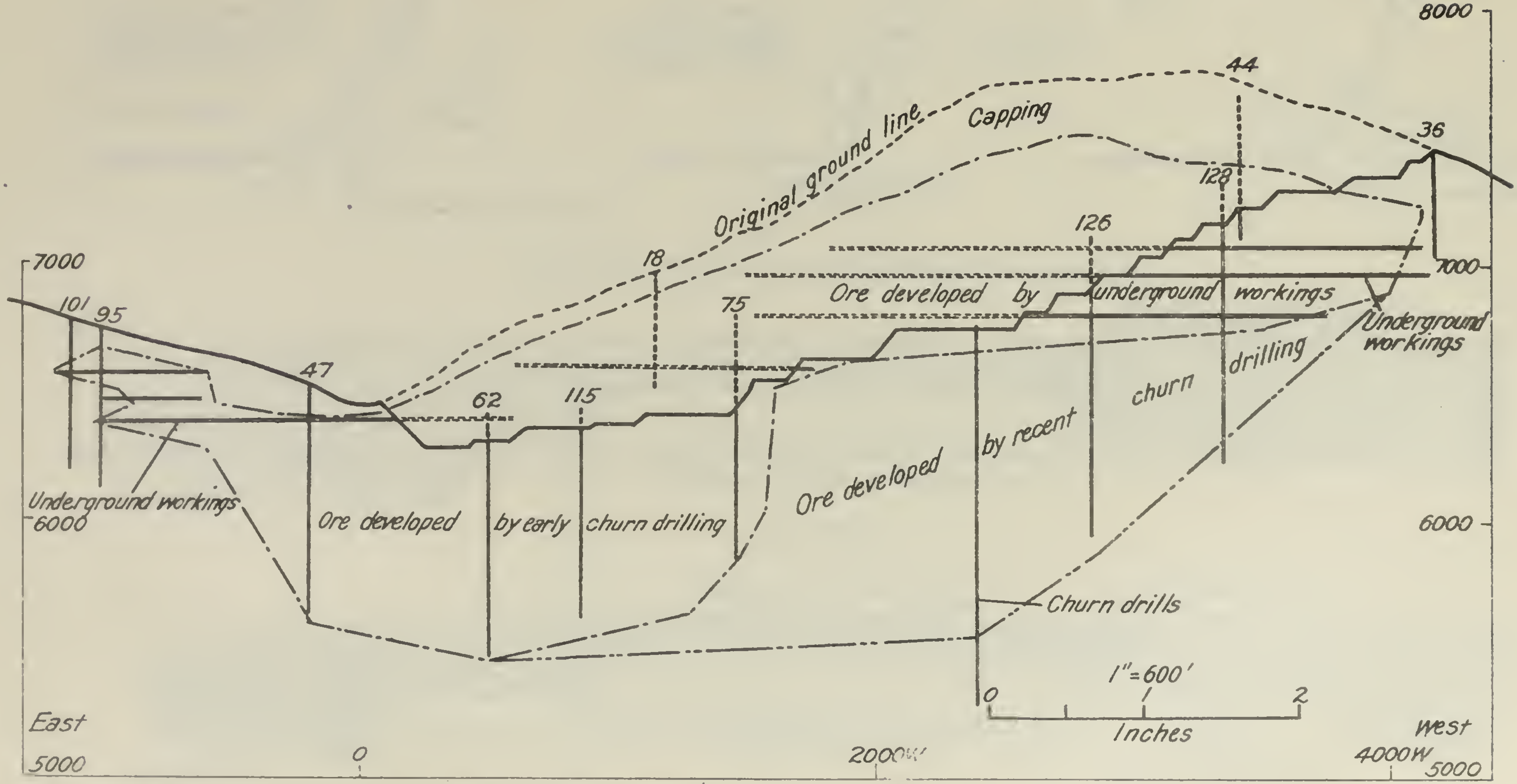


Figure 32. Illustrative section showing comparative ore development by underground workings and churn drills

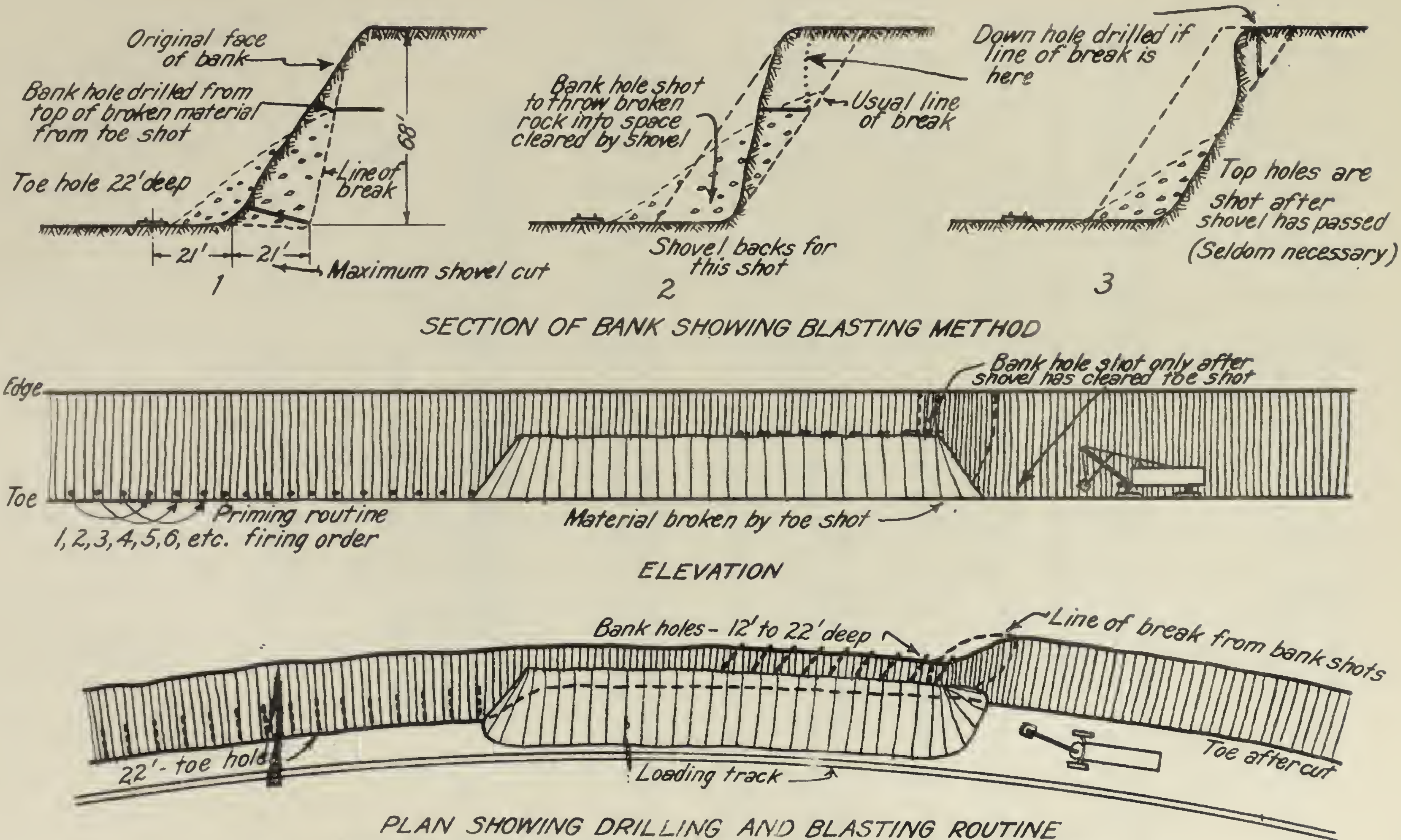


Figure 33 - Drilling and blasting practice Utah Copper Mine

4. Low idle or shutdown expense during periods of slack markets.

5. Safety in operation, the safety records being usually better than for most of the underground methods.

Disadvantages.-

1. Where much overburden has to be removed prior to extraction of ore, development expense, including interest on development capital prior to the productive stage, is high.

2. In severe weather, opencut work has obvious disadvantages.

3. Necessity for preventing flooding of the workings during heavy rains and spring thaws.

4. Necessity for sometimes removing large bodies of waste material within the pit area to get at ore lying beneath.

5. In deep pits a difficult problem is often presented in keeping the proper slopes and benches or berms on the sides of the pits to prevent slides and at the same time to remove no more barren material than is necessary.

6. Deep pits must have sufficient horizontal extent to permit reasonable haulage grades.

Results.- Two tables are given; one to show for the different ores, the metal yield and man-hours consumed per ton mined by the opencut method at mines that produced over $100,000 in 1929; the other gives operating costs.

Metal yield and man-hours consumed per ton mined by the opencut method at mines producing over $100,000 in 1929

Metal	Number of mines	Tons mined	Yield per ton	Man-hours per ton
Gold	0	0	0	0
Copper	7	31,563,645	20.6 lbs.	0.29
Lead-zinc	0	0	0	0
Iron	48	35,609,878	.475 ton	.39

Operating costs

Mine	Drilling and blasting	Shovel operations	Transportation	Miscellaneous	Total	Labor	Power and supplies	Material handled, 1928	Refer I.C. list, p. 36
Utah Copper:									
Stripping, per cu. yd.	$0.060	$0.042	$0.145	$0.027	10.274	$0.127	$0.147	7,220,034 yds. waste	(110)
Mining, per ton of ore	0.027	0.019	0.054	0.018	0.117	0.043	0.075	16,558,500 tons ore	
Total, per ton of ore	0.054	0.037	0.117	0.029	0.237	0.100	0.139		
United Verde:									
Stripping, per cu. yd.	0.293	0.161	0.198	0.018	0.670	0.304 (est.)	0.118	322,538 yds. waste	(111)
Mining, per ton of ore	0.139	0.077	0.086	–	0.302	0.167 (est.)	0.065	731,488 tons ore	
Total, per ton of ore	0.258	0.145	0.160	0.008	0.571	0.301	0.117		
Chino (steam):									
Stripping, per cu. yd.	0.092	0.193	0.284	0.049	0.633	0.121	0.154	1,577,373 yds. waste	(112)
Mining, per ton of ore	0.051	0.079	0.131	0.017	0.278	0.052	0.067	1,335,476 tons ore	
Total, per ton of ore	0.155	0.301	0.458	0.074	1.025	0.195	0.148		
Chino (electric):									
Stripping, per cu. yd.	0.097	0.078	0.219	0.030	0.424	0.068	0.092	2,976,357 yds. waste	(112)
Mining, per ton of ore	0.049	0.036	0.108	0.014	0.207	0.037	0.048	2,621,340 tons ore	
Total, per ton of ore	0.158	0.125	0.357	0.048	0.688	0.114	0.092		
Weighted averages:									
Stripping, per cu. yd.	0.079	0.073	0.180	0.030	0.368	0.117	0.134	12,096,302 yds. waste	
Mining, per ton of ore	0.035	0.027	0.066	0.017	0.145	0.047	0.070	21,246,804 tons ore	
Total, per ton of ore	0.080	0.068	0.169	0.034	0.354	0.113	0.147		

RECENT PROGRESS IN MINING PRACTICE

Improvement in recent years in underground mining practice has been in three principal directions: In the application of the mining method best suited to the individual conditions; in increased use of mechanical and electrical equipment; and in better organization underground so as to get greater efficiency per man and per machine.

Illustrative of the first are numerous recent changes from shrinkage stoping to cut-and-fill stoping, changes from shrinkage to sublevel stoping, from square-setting to top slicing, from top slicing to sublevel caving, in each case better results being achieved due to substituting a method better suited to local conditions.

Not only have radical changes been made by substitution of an entirely different mining method, but minor variations of mining methods and practice have resulted in safer and more economical operation.

The causes to which these changes in mining methods may be attributed are: Increased depth of workings; increased size of orebodies worked; reduction in assay value of ore; demand for increased output; increased cost of labor and supplies; and decrease in market value of products.

These same reasons will be responsible for future improvements in mining methods and practice, as the necessity for lower mining costs increases.

As an aid to the operator in working out his problems, the United States Bureau of Mines is disseminating descriptive reports and data, prepared by officials of the principal metal mines in the United States, and published as Information Circulars for free distribution. (See list at end of this paper.)

PRESENT TENDENCY TOWARD IMPROVEMENTS IN MINING METHODS

Much improvement has been made in mining methods during the past decade, but greater progress should be made during the next 10 years. Why? Because present low metal prices will not permit certain mines to continue operations except with improved practice and with the introduction of methods better suited to special conditions. Improved practice should include better underground organization and the training of workmen to greater efficiency.

Progress in mining methods will probably be in the direction of eliminating filling with waste, the decreased use of square-sets, and greater application of the retreating system of stoping. The tendency is toward the vertical alignment of stopes rather than in the direction of horizontal development. A system may be developed eventually by which large orebodies may be mined in vertical slices, starting from the extreme ends or sides of the deposit and retreating laterally, and from lower levels upwards, rather than by the usual method of mining horizontal slices starting at the upper levels and near the shaft. Where the walls are fairly solid, vertical slices could be mined by the sublevel stoping method; and where the walls are weak a method something like the top-slice method might be developed. There are places where such a method may be impracticable, but a start is already being made in this direction, and engineers should consider seriously such a method in planning future operations.

Another reason for a change in present methods at some mines is the fact that they must be worked deeper to develop new ore reserves, which will increase mining costs unless some radical improvement is made in present practice.

METHOD OF FIRST DEVELOPING AND MINING UPPER LEVELS

The more common practice is to open and mine the upper levels first - often, it is true, to work several levels at the same time - and to maintain production by sinking the shaft deeper from time to time and opening lower levels.

Among the chief reasons for the adoption of this method may be mentioned: Lack of knowledge regarding the deposits at depth and in lateral extent; necessity for speed in

opening the mine and bringing it into profitable production; need for using a relatively small capital outlay prior to profitable operation. Many companies do not have capital sufficient for exploitation in any other manner.

Some of the disadvantages of this more common practice which have brought about the adoption of the opposite procedure of starting at the bottom and outer limits of the deposit are as follows:

First, high rock pressures often develop on the lower levels. As mining operations progress downward, the maintenance of hoisting shafts, airways, and drifts adjacent to worked-out ground becomes an increasing item of expense, and during mining operations workings often require increased support and in some cases are difficult to support.

Second, where stopes are filled concurrently with the mining of the ore, or subsequently, the waste from development on the lower levels is usually relied upon to furnish a considerable portion of the fill. This entails hoisting of waste from lower to higher levels, often interferes with hoisting and tramming of ore, and may mean added expense on several levels for separate ore and waste pockets, for hoisting equipment, and perhaps even for larger shafts.

Third, breaks to the surface produced by ground movement early in the life of the operation may entail heavy pumping expense over the entire life of the mine, due to entrance of surface waters.

Fourth, the ore itself may be wet, and when mining from the top downward drainage will be poor and the mine will be wet on all levels.

Fifth, by this method, toward the end of the life of the mine the costs are apt to be high. One of the companies which is planning the reverse order of mining is at present suffering from high costs at one of its mines which is in its old age. When a mine has just been developed to the productive stage, all the early conditions are generally at their best for securing low mining costs. Ground movement and subsidence have not started, and problems of support are less serious than later.

In mining from the top downward, hoisting costs are at their lowest during the early life of the mine, the flow of water may or may not be minimum but the static head against which the water must be pumped is constantly increasing. Often as mining progresses to successively lower levels, these problems become increasingly difficult of solution. Thus, considerably before the orebody is all mined, the cost of production may reach the point where mining can no longer be profitably done.

METHOD OF FIRST DEVELOPING AND MINING LOWER LEVELS

Several instances have been noted where development and mining have been or are to be started at the lowest levels on the orebody or on a block of considerable vertical depth, followed by development and mining on successively higher levels.

By mining the lower levels first, troubles incident to ground movement, caving, and subsidence will not interfere with active mining operations. Also, waste from development of upper levels can always be disposed of readily and cheaply by gravity transfer, and

in the event of a heavy inrush of water, or serious breakdown of pumping equipment or source of power, the worked-out lower levels will furnish a large storage reservoir.

In the bottom levels operating difficulties may arise, first in wide orebodies if the ore is mined by wide horizontal slices. Where large areas are mined and filled, the filling can not be expected to offer sufficient support to prevent major subsidence, if the pressures involved are so great that movement can take place by the squeezing of the fill. Once large-scale movement starts, breaking and crushing of ore above the lower worked-out areas is bound to occur, often making mining on the upper levels difficult and expensive. This can be prevented by beginning at the end or margins of the orebody, or by starting sections or blocks and distributing the stoping along a roughly vertical plane, rather than over wide horizontal areas. In this way mining always would be retreating from areas of possible subsidence and crushing.

BUREAU OF MINES INFORMATION CIRCULARS IN MINING PRACTICE
LISTED ACCORDING TO PRINCIPAL STOPING METHODS

Open Stoping

Ref.	I.C.	
(1)	6092	Cummings, A. M. Method and Cost of Mining Magnetite in the Mineville District, N. Y.
(2)	6113	Netzeband, Wm. F. Method and Cost of Mining Zinc and Lead at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Okla.
(3)	6121	Netzeband, Wm. F. Method and Cost of Mining Zinc and Lead at Mine No. 2, Tri-State District, Picher, Okla.
(4)	6138	Eaton, Lucien. Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan.
(5)	6149	McNaughton, C. H. Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.
(6)	6150	Banks, Leon M. Mining Methods and Costs in the Waco District.
(7)	6159	Keener, Oliver W. Method and Cost of Mining at Barr Mine, Tri-State Lead and Zinc District.
(8)	6160	Poston, Roy H. Method and Cost of Mining at No. 8 Mine, St. Louis Smelting and Refining Company, S. E. Missouri District.
(9)	6170	Jackson, Chas. F. Methods of Mining Disseminated Lead Ore at a Mine in the S. E. Missouri District.
(10)	6174	Netzeband, Wm. F. Method and Cost of Mining Zinc and Lead at No. 3 Mine, Tri-State District, Crestline, Kans.
(11)	6179	Eaton, Lucien. Mining Soft Hematite at Mine No. 2 of the Marquette Range, Michigan.
(12)	6180	Eaton, Lucien. Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Michigan.
(13)	6193	Jackson, Chas. F. Mining Ore in Open Stopes, Central and Eastern United States.
(14)	6239	Coy, Harley A. Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.
(15)	6286	Keener, Oliver W. Methods and Costs of Mining at the Hartley-Grantham Mine, Tri-State Zinc and Lead District.
(16)	6361	Kniffin, Lloyd M. Mining and Engineering Methods and Costs of the Hanover Bessemer Iron and Copper Co., Fierro, N. Mex.
(17)	6369	Schaus, O. M. Mining Methods and Costs at the Montreal Mine, Montreal, Wis.
(18)	6397	Kegler, Vern L. Mining Methods of the Ducktown Chemical and Iron Co., Mary Mine, Isabella, Tenn.
(19)	6402	Pierce, A. L. Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.
(20)	Bull. 306	Crane, W. R. Mining Methods and Practice in the Michigan Copper Mines.
(21)		Howbert, V. D., and Bosuston, R. Mining Methods and Costs at Presidio Mine of the American Metal Co. of Texas. A.I.M.E., T. P. 334, 1930, 15 pp.

Shrinkage Stoping

(26)	6186	Bradley, P. R. Mining Methods and Costs, Alaska Juneau Gold Mining Co., Juneau, Alaska.

(27) 6260 Nelson, W. I. Mining Methods and Costs at Engels Mine, Plumas County, Calif.

(28) 6276 Elmer, Wm. W. Mining Methods and Costs at the Black Butte Quicksilver Mine, Lane County, Oreg.

(29) 6284 Heizer, O. F. Method and Cost of Mining Tungston Ore at the Nevada-Massachusetts Mines at Mill City, Nev.

(30) 6293 Jackson, Chas. F. Shrinkage Stoping.

(31) 6294 Reeder, Edwin D. Method and Cost of Mining Fluorspar at Rosiclare, Ill.

(32) 6322 Henry, R. J. Mining Methods and Costs at the Teck-Hughes Gold Mines, Ltd., Kirkland Lake, Ontario.

(33) 6327 Hezzelwood, George W. Mining Methods and Costs at the Consolidated Cortez Silver Mine, Cortez, Nev.

(34) 6384 Cronk, A. H. Mining Methods of the Rosiclare Lead and Fluorspar Mining Co., Rosiclare, Ill.

(35) 6390 Graff, W. W. Mining Practices, Methods, and Costs at Mine No. 4 of the Marquette Range, Michigan.

(36) 6413 Youtz, Ralph B. Mining Methods at the Eighty Five Mine, Calumet and Arizona Mining Co., Valedon, N. Mex.

(37) 6464 Dickson, R. H. Methods and Costs of Mining Copper Ore at the Verde Central Mines, Inc., Jerome, Ariz.

Cut-and-Fill Stoping

(45) 6145 Richert, George L. Mining Methods at Minas de Matahambre, Matahambre, Pinar del Rio, Cuba.

(46) 6247 Catron, William. Mining Methods, Practices and Costs of the Cananea Consolidated Copper Co., Cananea, Sonora, Mexico.

(47) 6289 Lavender, H. M. Mining Methods at the Campbell Mine of the Calumet and Arizona Mining Co., Warren, Ariz.

(48) 6307 Leland, Everard. Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico.

(49) 6368 Matson, J. T., and Hoag, C. Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico.

(50) 6440 Quayle, T. W. Mining Methods and Practices at the United Verde Copper Mines, Jerome, Ariz.

(51) 6416 Vanderburg, W. O. Mining Methods at the Block P Mine of the St. Joseph Lead Co., Hughesville, Mont.

Square-Set Stoping

(55) 6168 Snow, F. W. Mining Methods and Costs at the Magma Mine, Superior, Ariz.

(56) 6250 D'Arcy, Richard L. Mining Practices and Methods at the United Verde Extension Mining Co., Jerome, Ariz.

(57) 6290 Hewitt, E. A. Mining Methods and Costs at the Park Utah Mine, Park City, Utah.

(58) 6311 Vanderburg, W. O. Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.

(59) 6360 Wade, James W. Mining Methods and Costs at the Tintic Standard Mine, Tintic District, Utah.

Ref. I.C.

(60) 6370 McGilvra, D. B., Healy, A. J. Methods of Mining at the Black Rock Mine, Butte and Superior Mining Co., Butte, Mont.

(61) 6371 Dailey, M. J. Methods and Costs at the Silver King Coalition Mines Co., Park City, Utah.

(62) 6372 Berg, J. E. Mining Methods at the Page Mine of the Federal Mining and Smelting Co., Page, Idaho.

(63) 6377 Richard, F. W. Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.

(64) 6407 Brown, U. E. Mining Methods of the Bunker Hill and Sullivan Mining and Concentrating Co., Kellogg, Idaho. (See also reference 46.)

Stull and Post Sets and Fill

(70) 6232 Foreman, C. H. Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho.

(71) 6238 Wethered, C. E., and Coady, Leo J. Mining Methods at the Morning Mine of the Federal Mining and Smelting Co., Mullan, Idaho.

Top Slicing

(75) 6325 Haselton, W. D. Underground Mining Practices and Costs at a Mesabi Range (Minn.) Mine Using the Top-Slicing System.

(76) 6380 Graff, W. W. Mining Practices, Methods and Costs at Mine No. 5 of the Marquette Range, Michigan.

(77) 6410 Jackson, Chas. F. Mining by the Top-Slicing Method, with some Notes on Sublevel Caving. (See also reference 46.)

(11) 6179 Eaton, Lucien. Mining Soft Hematite at Mine No. 2 of the Marquette Range, Michigan.

Sublevel Caving

(90) 6348 Schaus, O. M. Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.

(17) 6369 Schaus, O. M. Mining Methods and Costs at the Montreal Mine, Montreal, Wis. (See also reference 77.)

Block Caving

(100) 6107 Mosier, McHenry, and Sherman, Gerald. Mining Practice at Morenci Branch, Phelps Dodge Corp., Morenci, Ariz.

(101) 6167 Thomas, Robert W. Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.

(102) 6169 Stoddard, Alfred C. Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.

Ref. I.C.

(103) 6350 Gardner, E. D. Undercut Block-Caving Method of Mining in Western Copper Mines.

(104) MacLennan, F. W. Miami Copper Company Method of Mining Low-Grade Orebody. Trans. Am. Inst. Min. and Met. Eng., 1930, pp. 39-86.

(105) Greninger, I. L. Mine Development and Underground Construction of Andes Copper Mining Co. at Potrerillos, Chile. Trans. Am. Inst. Min. and Met. Eng., 1929, pp. 144-166. (See also reference 46.)

Opencut Methods

(110) 6234 Soderburg, Adolph. Mining Methods and Costs at the Utah Copper Co., Bingham Canyon, Utah.

(111) 6248 Alenius, E. M. J. Methods and Costs of Stripping and Mining at the United Verde Open-Pit Mine, Jerome, Ariz.

(112) 6412 Thorne, H. A. Mining Practice at the Chino Mines, Nevada Consolidated Copper Co., Santa Rita, N. Mex.

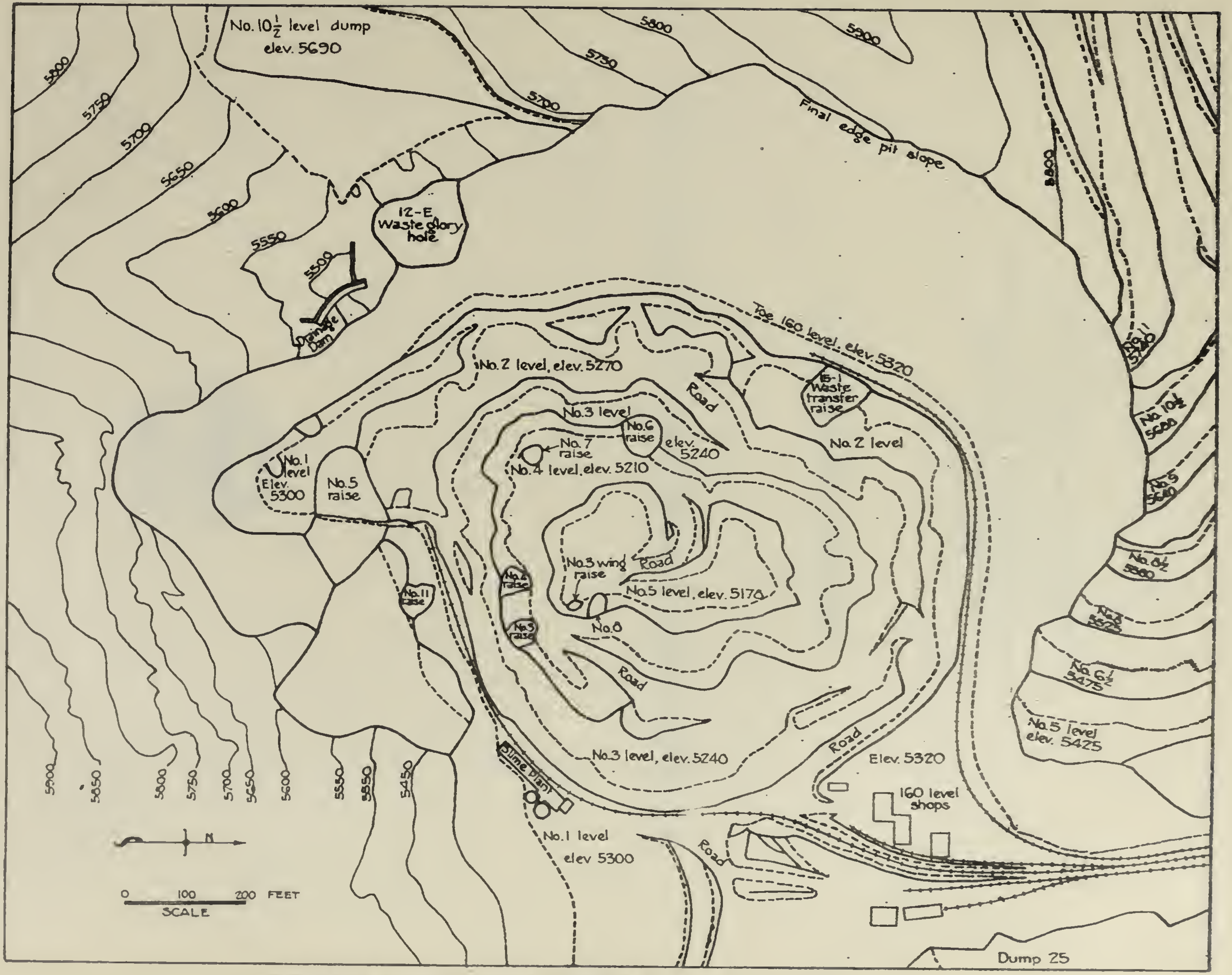


FIGURE 34 PLAN OF UNITED VERDE PIT, JULY 1, 1929

I. C. 6504

September, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

UMBER, SIENNA, AND OTHER BROWN EARTH PIGMENTS

BY

R. M. Santmyers

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

UMBER, SIENNA, AND OTHER BROWN EARTH PIGMENTS[1]

By R. M. Santmyers[2]

CONTENTS

FOREWORD

Ocher and ochery earths, the most important of the naturally occurring earth pigments, have been covered in United States Bureau of Mines Information Circular 6132.

The present paper deals primarily with umber and sienna, together with other natural brown earth pigments, such as Vandyke brown or Cassel earth, Cappagh brown, Caledonian brown, Cologne earth, and mineral brown.

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1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6504."

2 Mineral specialist, rare metals and nonmetals division, U. S. Bureau of Mines.

GENERAL DESCRIPTION

Umber and sienna, like ocher, are naturally occurring mineral pigments composed largely of clay permeated with hydrated iron (ferric oxide), but they differ from ocher in that they also contain hydrated manganese oxide. The presence of manganese accounts for their browner color. In addition to silica and alumina (clay) and the coloring oxides of iron and manganese, these pigments often contain lime, barium, and other metallic salts as impurities.

Sienna most nearly approaches ocher in chemical composition, and the chief distinction is its physical character. It grades into ocher with decreasing iron content and into umber with increasing manganese content. Sienna is translucent rather than opaque and is more of a stain than a pigment; as the iron oxide content is rather high (often 60 to 80 per cent), the color is strong and rich and varies from pure brown to reddish brown. This pigment is known under several names: raw sienna, terra di Siena, and Italian earth.

Burnt sienna is a pigment of an orange-brown tint made by cautionsly calcining raw sienna. It is divided into two grades; American, and Italian burnt sienna.

American burnt sienna is a reddish-brown pigment containing from 25 to 60 per cent iron oxide (Fe_2O_3). One grade, found in Pennsylvania, analyses as high as 80 per cent and is known as double strength sienna.

Italian burnt sienna is a pigment generally containing more iron than the American product, averaging 60 to 75 per cent iron oxide.[3] American pigments of the same iron content differ materially in shade, having a Havana-brown color, whereas the Italian is more of a maroon. Italian siennas when tinted with 20 per cent of white, show a bluish tint, whereas American siennas yield a brownish or yellowish hue.

Raw umber, also known as Turkey umber, Levant umber, and terra ombra, is a greenish to yellowish brown siliceous and ferruginous earth, containing a considerable proportion of one or other of the higher oxides of manganese, Mn_3O_4 and MnO_2.[4] It is similar in many respects to raw sienna except that it is much coarser and contains more manganese oxide. All umbers contain over 5 per cent of manganese dioxide. Like sienna, umber is a permanent pigment, not being injured by light or by impure atmospheres.

Burnt umber (velvet brown, chestnut brown, etc.), results from the calcination of raw umber. The color is much richer and warmer than that of the raw umber. The colors are equally permanent and can be used in conjunction with all other durable pigments.

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3 U. S. Tariff Commission, Tariff Information Survey, A-15, 1922, p. 79.
4 Thorpe, E., A Dictionary of Applied Chemistry, London, vol. 5, 1924, p. 298.

Turkey umbers usually contain more manganese and iron than the American product. Umbers rich in manganese when calcined yield brighter shades in proportion to their fineness as determined either by grinding or levigation.

Other naturally occurring earth pigments are Cappagh brown, Caledonian brown, Vandyke brown, (Cassel earth), Cologne earth, and metallic or mineral brown.

Cappagh brown, also known as mineral brown, euchrome, etc., is a highly manganiferous and ferruginous earth obtained from the Cappagh mine, near Skibbereen, Ireland. When heated to 100° C. it loses water and assumes a fine reddish-brown hue like that of burnt sienna.

Caledonian brown possesses a reddish tint, altered upon ignition almost to black. This earth is composed chiefly of manganese and iron oxides and hydroxides.

Vandyke brown, also known as Cassel brown, Cassel earth, and Rubens brown, are terms applied more or less indiscriminately to pigments composed of clay, iron oxides, decomposed wood, peat, lignite, and other organic matter. These pigments are obtained from bog earth, peat, or lignite deposits and are essentially mixtures of clay or ocher with varying proportions of bituminous matter. Though of a fine rich hue, this variety of Vandyke brown is decidedly fugitive, especially in water color. Certain shades of umbers and siennas are called Vandyke browns.

Cologne earth is a bituminous (lignitic) Vandyke brown which has been gently roasted, in order to make the color darker and more permanent.

Metallic brown or mineral brown is made by calcining an impure iron oxide (limonitic iron ore). It usually contains from 40 to 75 per cent ferric oxide, averaging 50 per cent. It is variable in color but resembles burnt sienna.

PROPERTIES

All pigments consist of three parts: (1) The principal color-producing ingredient, which is usually hydrous iron oxide in brown earth pigments; (2) the secondary or modifying coloring ingredient, such as manganese for umbers and carbonaceous matter for brown and blacks; and (3) the base, filler, dilutent, or carrier of the color.[5]

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5 Wilson, Hewitt, Ochers and Mineral Pigments of the Pacific Northwest; Bull. 304, Bureau of Mines, 1929, pp. 10-11.

The base in practically all naturally occurring earth pigments is clay. Clay is the weathered product of the silicate rocks, which contains sufficient hydrous silicate of alumina in the softened condition to produce a plastic or semiplastic mass when tempered with water. The composition of clay varies widely, hence the base compositions of most ochers, umbers, sienna, and other earth pigments may be quite different. This base composition, unlike that in artificial pigments, can only be changed or controlled by (1) selection or blending of varying portions of the deposit and (2) by removal of some impurities by simple purification methods.

USES

Umber, sienna, and other naturally occurring earth pigments containing the oxides of iron and manganese have, with the exception of color, much the same physical properties as ocher and are similarly used in industry.

There are two principal uses for umber and sienna - as a pigment in paints and for wood stains. As paint pigments, they are very permanent, have good covering power, mix well with all other durable pigments without alteration, are not affected by acids or alkalies, and are relatively inexpensive. As coloring agents, they are also used in linoleum, oil cloth, paper, etc., and to a minor extent in the ceramic industry.

Ground sienna is used in comparatively large amounts for lithographic and typographic printing and as an artist's color - either alone or as a base for lake pigments upon which organic coloring matters are precipitated from solution.[6] A popular lake of this type is mahogany (cashew) lake used for lithographic and letter-press printing. It is admirably suited for this purpose, as the lake dries very quickly. Sienna is translucent rather than opaque, and upon this characteristic is based its large use as a stain and colored wood filler for wood that is to be shellacked or varnished. In mixed paints, siennas are used for their tinting quality, the resulting shade being a yellowish maroon or salmon color. In the preparation of these paints, however, it is customary to mix two or more pigments together in order to produce a paint with the specified color and properties desired.

Umber has similar uses in paints but has a greater covering power than sienna. It yields very durable varnish paints that dry quickly and hard. Umber is an essential ingredient of weatherproof paints and gives an unusual variety of colors for house painting. A considerable quantity of umber is consumed in the wall paper industry to produce the ground shade.

American umbers and siennas are used in the most part as pigments and stains in the less expensive paints and color varnishes. However, umbers and siennas produced in Pennsylvania are used in all grades of paints, as the deposits are favorably situated in relation to consuming centers.

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6 U. S. Tariff Commission, Work cited, pp. 79-80.

The other earth pigments containing iron and manganese, such as Cappagh brown, Caledonian brown, Vandyke brown, and metallic brown are used in the most part as substitutes for umbers and siennas, though those pigments may be classed as umbers and siennas containing varying amounts of iron and other oxides.

Vandyke or Cassel brown is not important as an oil color, as the paints dry slowly and has no great covering power, but it has a good body, is fairly permanent, and mixes well with other colors. The largest consumption is for wall and fancy paper making, for coloring cardboard and for the manufacture of wood stains.

SUBSTITUTES

Umber and sienna have no substitutes in the sense of similar cheaper products. While sienna is the most expensive of the common brown pigments the range in quality is such that the lower grades are cheaper than many of the pigments that could be classed as substitutes. Of these Vandyke or Cassel brown - essentially a mixture of clay or ocher with varying proportions of bituminous matter - is probably the most important.[7] Two other browns, more permanent in color than Vandyke brown but far less richly tinted, sold under the same name, consist of a brown variety of caput mortuum and a calcined ferruginous ocher, respectively. Imitation Vandyke browns are made, consisting of ocher, colcothar, and lampblack, in admixture. Other substitutes for Vandyke browns are made artificially from charred organic substances, such as bark, cork cuttings and bone dust. The various iron oxides, such as Spanish brown and colcothar, and artificial browns produced by chemical precipitation are also used. Brown lakes, except those containing umber, sienna, or ocher, are not used to any considerable extent as substitutes for these important pigments.

SOURCES OF SUPPLY

Little information has been published recently regarding the deposits of mineral pigments in the United States.

Neither umber nor sienna are produced in large amounts in this country, chiefly for the reason that the products of certain foreign countries have established an international reputation for quality.

In the United States, deposits of umber have been worked near Bethlehem, Doylestown, and Bethel in Pennsylvania, while sienna has been produced near Reading, Pa., and at Valley Station near West End, N. J. Umber could be produced also in the ocher and manganese district in the vicinity of Cartersville, Ga., if the demand existed.[8] Although many of the pigments produced in this country are called "ochers," their darker color and higher iron content should place them in the sienna class when they are compared with the lighter colored French ochers.

7 Ladoo, R. B., Non-Metallic Minerals, Occurrence, Preparation, Utilization: New York, 1925, p. 373.

8 Wilson, Hewitt, Ochers and Mineral Pigments of the Pacific Northwest: Bull. 304, Bureau of Mines, 1929, p. 7.

Umber is produced in practically all countries, but the finest qualities are found in Cyprus, Sicily, Asia Minor, and the Netherlands. Turkish umber, chiefly from the island of Cyprus, is the standard. Italian sienna, produced in Tuscany, has the same preeminence in its field as Turkish umber or French ocher in the world markets. Germany is the only other important source of supply, sienna being produced along with considerable quantities of umber in the Harz Mountain district.

The other brown earth pigments are likewise produced in several countries.

It is reported that the original supplies of Cappagh brown, obtained from the Cappagh mine, near Skibbereen, Ireland, and Caledonian brown are exhausted. Substitutes for these pigments, consisting of a mixture of Vandyke brown and burnt sienna, however, are still sold under the same names.

The chief source of Vandyke or Cassel brown has been Germany, though it has been reported that a very acceptable type of this pigment occurs in abundance 6 miles east of Hanna, Wyo.

According to Ladoo:[9] "Similar material is also reported about 15 miles northwest of Putnam, N. Mex., and 8 miles south of Box Elder, Mont."

METHODS OF PREPARATION

The mining and preparation of umber and sienna do not differ essentially from the mining and preparation of ocher, which are described in Bureau of Mines Information Circular 6132. The following brief outline of the Italian method is taken from an unpublished consular report.

> During the warm seasons the raw sienna earth is excavated, after which it is usually washed in tanks, and the sienna, containing perhaps 70 to 80 per cent of water is banked in small piles and left in the sun to dry. Burning the sienna is extensively carried on at the place of excavation, but the major portion is shipped to Leghorn in the raw state and there treated.[10]

Processes of drying are extremely simple. Such of the raw material as is to be ground unburnt is first dried in kilns at a low heat, until about only 5 per cent of moisture remains. The material to be burnt is placed in a brickkiln under extreme heat. Charcoal is generally the fuel used. Buhrstone mills are used for final grinding.

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9 Ladoo, R. B., Work cited, p. 373.

10 Jackson, Jesse B., Sienna Earth - Production and Exportation: Consular Rept. 243515, Leghorn, Italy, March 31, 1927.

DOMESTIC PRODUCTION

The requirements of the paint producers in the United States for umber and sienna are met primarily by imports. Neither of these pigments is produced from domestic sources in any large amounts. Production statistics were gathered by the United States Geological Survey for several years but were discontinued in 1914. On account of the small production, figures for umber and sienna were combined. There is no known domestic production of natural Vandyke or Cassel brown.

Table 1 shows the production of umber and sienna separately in the United States for the years 1896 to 1902, while Table 2 shows the total production of these two pigments from 1903 to 1914, the last year for which official figures are available.

Table 3 shows the production of ocher, umber, sienna, and other iron-oxide pigments in the United States for the census years, 1919 to 1929.

Table 1.- Umber and sienna produced in the United States, 1896-1902 1/

Year	Umber, short tons	Value	Sienna, short tons	Value
1896	165	$ 2,646	395	$ 5,416
1897	2/ 1,080	11,710	620	10,610
1898	3/ 1,177	8,285	689	11,140
1899	473	4,151	688	8,205
1900	1,452	26,927	957	14,771
1901	759	11,326	305	9,304
1902	480	11,230	189	4,316

1/ Mineral Paints: Mineral Resources of the United States, Part II, Nonmetals, U. S. Geol. Survey (Annual).
2/ Includes 600 tons of Spanish brown from Maryland.
3/ Includes 640 tons of Spanish brown from Maryland.

Table 2.- Umber and sienna produced in the United States, 1903-1914 1/

Year	Umber and sienna		Year	Umber and sienna	
	Short tons	Value		Short tons	Value
1903	666	$15,367	1909	1,546	$43,872
1904	522	12,960	1910	1,015	26,700
1905	689	17,004	1911	1,005	26,225
1906	657	17,394	1912	805	21,975
1907	730	19,309	1913	776	20,790
1908	2/ 2,756	70,996	1914	790	21,070

1/ Mineral Paints: Mineral Resources of the United States, Part II, Nonmetals, U. S. Geol. Survey (Annual).
2/ Exceptional output (1,452 tons) of umber.

Table 3.- Ocher, umber, sienna, and other iron-oxide pigments produced in the United States 1/

Year	Pounds	Value
1919	202,181,787	$ 3,778,942
1921	82,757,631	1,737,478
1923	122,000,759	2,718,088
1925	57,063,165	1,285,294
1927	108,357,735	3,357,895
1929 2/	106,060,392	3,381,280

1/Census of Manufactures, Bureau of the Census (biannual).
2/Preliminary.

IMPORTS

Tables 4, 5, 6, 7, 8, and 9, show the imports for consumption of umber, sienna, Vandyke brown, and "all other mineral earth pigments."

The imports of umber and sienna, ground in oil or water, were not large prior to the World War. The imports of umber in oil or water, however, have increased considerably in recent years, and while figures for the same classification for sienna are not reported separately, they too, no doubt, have increased.

Both umber and sienna have been imported largely in crude form, but in recent years, the imports of powdered sienna have exceeded those of crude sienna.

In the case of Vandyke or Cassel brown, there has been a gradual increase in imports, although in 1930, due perhaps to general trade conditions, the imports of this pigment took a decided slump.

The imports of "all other mineral-earth pigments" have shown a downward trend, amounting to only 3,441 pounds in 1930 as compared with 1,325,168 pounds in 1923.

Table 4.-Imports for consumption of umber into the United States, 1901-1922[1]

Year	Dry		Ground in oil or water		Total	
	Pounds	Value	Pounds	Value	Pounds	Value
1901	1,562,247	- -	3,184	- -	1,565,431	$ 12,510
1902	1,887,426	- -	11,999	- -	1,899,425	16,133
1903	2,159,914	$ 17,685	9,656	$ 587	2,169,570	18,272
1904	2,261,793	19,727	13,133	784	2,274,926	20,511
1905	2,580,501	20,763	6,783	461	2,587,284	21,224
1906	2,948,539	23,732	6,028	418	2,954,567	24,150
1907	3,395,690	26,502	2,569	211	3,398,259	26,713
1908	2,391,153	19,461	15,556	803	2,406,709	20,264
1909	3,104,037	26,125	4,953	256	3,108,990	26,381
1910	3,994,286	28,819	11,813	734	4,006,099	29,553
1911	3,163,614	22,025	751	87	3,164,365	22,115
1912	4,857,706	31,408	3,179	218	4,860,885	31,626
1913	5,230,447	36,397	6,042	374	5,236,489	36,771
1914	2/	2/	2/	2/	7,886,716	45,280
1915	2/	2/	2/	2/	5,835,292	32,182
1916	2/	2/	2/	2/	10,405,923	69,470
1917	2/	2/	2/	2/	4,334,394	42,620
1918	2/	2/	2/	2/	487,623	13,895
1919	2/	2/	2/	2/	2,315,377	23,244
1920	2/	2/	2/	2/	7,643,235	85,424
1921	2/	2/	2/	2/	3,591,339	45,720
1922	2/	2/	2/	2/	4,927,957	55,554

1/ Foreign Commerce and Navigation of the United States (Annual); Bureau of Foreign and Domestic Commerce.

2/ Not separately classified.

Table 5.- Imports for consumption of umber into the United States, 1923-1930 [1]

Year	Crude not ground		Washed or ground		Total	
	Pounds	Value	Pounds	Value	Pounds	Value
1923	7,386,658	$ 61,690	1,303,301	$ 29,433	8,689,959	$ 91,123
1924	6,895,473	58,303	1,258,694	27,134	8,154,167	85,437
1925	7,587,224	56,213	921,162	22,719	8,508,386	78,932
1926	5,775,040	36,688	1,324,283	48,106	7,099,323	84,794
1927	7,038,761	44,426	1,236,469	30,092	8,275,230	74,518
1928	6,264,844	39,182	1,290,679	29,667	7,555,523	68,849
1929	7,083,100	43,977	1,189,110	28,144	8,272,210	72,121
1930	3,678,044	23,128	1,052,048	22,339	4,730,092	45,467

1/ Foreign Commerce and Navigation of the United States (Annual), Bureau of Foreign and Domestic Commerce.

Table 6.- Imports for consumption of sienna into the United States[1]

Year	Dry or crude		Ground in oil or water		Total	
	Pounds	Value	Pounds	Value	Pounds	Value
1901	1,106,553	$ 18,394	2/ 13,861	2/$ 1,004	1,120,414	$ 19,398
1902	1,534,878	27,299	2/ 5,921	2/ 494	1,540,799	27,793
1903	1,873,532	28,447	1,387	123	1,874,919	28,570
1904	1,286,301	22,118	5,770	396	1,292,071	22,514
1905	1,737,909	26,097	2,886	227	1,740,795	26,324
1906	1,941,664	32,673	- -	- -	1,941,664	32,673
1907	2,176,566	34,752	14,629	864	2,191,195	35,616
1908	1,756,273	28,407	7,621	458	1,763,894	28,865
1909	2,402,901	32,913	6,114	421	2,409,015	33,334
1910	3,048,203	46,866	6,233	453	3,054,426	47,319
1911	2,888,365	36,995	9,927	667	2,898,292	37,662
1912	3,056,064	45,354	6,021	440	3,062,085	45,794
1913	3,270,715	48,443	2,502	92	3,273,217	48,535
1914	7,815,323	63,958	2/	2/	7,815,323	63,958
1915	6,105,515	65,283	2/	2/	6,105,515	65,283
1916	4,537,999	82,999	2/	2/	4,537,999	82,999
1917	2,643,748	57,330	2/	2/	2,643,748	57,330
1918	1,641,214	60,154	2/	2/	1,641,214	60,154
1919	1,734,807	46,134	2/	2/	1,734,807	46,134
1920	5,414,477	145,480	2/	2/	5,414,477	145,480
1921	2,545,398	104,284	2/	2/	2,545,398	104,284
1922	2,721,433	105,532	2/	2/	2,721,433	105,532
1923	2,660,018	83,177	2/	2/	2,660,018	83,177
1924	1,501,490	42,473	2/	2/	1,501,490	42,473
1925	1,385,608	34,809	2/	2/	1,385,608	34,809
1926	2,018,289	49,432	2/	2/	2,018,289	49,432
1927	1,668,065	42,831	2/	2/	1,668,065	42,831
1928	1,591,935	50,013	2/	2/	1,591,935	50,013
1929	1,676,663	47,378	2/	2/	1,676,663	47,378
1930	901,879	34,747	2/	2/	901,879	34,747

1/ Foreign Commerce and Navigation of the United States (Annual), Bureau of Foreign and Domestic Commerce.

2/ Not separately classified.

Table 7.- Imports for consumption of sienna and ocher into the United States, 1922-1930 [1]

Year	Sienna and ocher washed or ground	
	Pounds	Value
1922	2/ 4,802,012	70,960
1923	17,470,629	238,338
1924	17,994,225	204,237
1925	18,651,893	238,258
1926	18,521,765	334,693
1927	18,920,745	395,684
1928	18,492,511	366,806
1929	18,979,131	343,111
1930	12,292,216	203,031

1/ Foreign Commerce and Navigation of the United States (Annual), Bureau of Foreign and Domestic Commerce.
2/ Sept. 22 to Dec. 31.

Table 8.- Vandyke brown, Cassel earth or Cassel brown imported for consumption into the United States [1]

Year	Pounds	Value
1911	- -	$ 491
1912	- -	3,192
1913	- -	2,187
1914	- -	1,302
1915	- -	1,208
1916	- -	790
1917	- -	2,759
1918	33,600	2,573
1919	70,002	2,416
1920	601,646	22,821
1921	213,278	5,334
1922	420,756	10,097
1923	412,014	10,252
1924	477,339	9,651
1925	734,618	12,783
1926	1,475,975	22,862
1927	1,108,031	19,334
1928	1,260,067	22,530
1929	1,363,097	24,581
1930	952,927	19,096

1/ Foreign Commerce and Navigation of the United States (Annual), Bureau of Foreign and Domestic Commerce.

Table 9.- "All other mineral-earth pigments" imported for consumption into the United States 1/

Year	Pounds	Value
1922 2/	34,350	$ 5,467
1923	1,325,168	53,340
1924	240,481	9,517
1925	235,442	8,716
1926	29,396	2,708
1927	257,885	2,553
1928	15,734	1,307
1929	6,610	1,786
1930	3,441	2,106

1/ Foreign Commerce and Navigation of the United States (Annual), Bureau of Foreign and Domestic Commerce.

2/ Sept. 22 to Dec. 31.

TARIFF HISTORY

Umber and sienna, together with ocher, were first mentioned specifically in the tariff act of 1883. Subsequently, except under the tariff act of 1894, they have been dutiable as shown in Table 10.

Table 10.- Tariff rates on ocher, umber and sienna

Act of-	Par.	Tariff classification or description	Rates of duty, specific and ad valorem
1883	---	Ocher and ochery earths, umber and umber earths, and sienna and sienna earths, when dry	½ cent per pound
		When ground in oil	1½ cents per pound
1890	54	Ocher and ochery earth, sienna and sienna earths, umber and umber earths, not specilly provided for in this act, dry	¾ cent per pound
		Ground in oil	1½ cents per pound
1894	42	Ocher and ochery earths, sienna and sienna earths, umber and umber earths, ground in oil	1¼ cents per pound
1894	566	Ocher and ochery earths, sienna and sienna earths, umber and umber earths, not specially provided for in this act, dry	Free.
1897	49	Ocher and ochery earths, sienna and sienna earths, umber and umber earths, not specially provided for, when crude or not powdered, washed, or pulverized	1/8 cent per pound
		If powdered, washed, or pulverized	3/8 cent per pound
		If ground in oil or water	1½ cents per pound

Table 10.- Tariff rates on ocher, umber and sienna - Cont'd

Act of-	Par.	Tariff classification or description	Rates of duty, specific and ad valorem
1909	47	Ocher and ochery earths, sienna and sienna earths, and umber and umber earths, not specially provided for in this section, when crude or not powdered, washed, or pulverized	1/8 cent per pound
		If powdered, washed, or pulverized	3/8 cent per pound
		If ground in oil or water	1 cent per pound
1913	55	Ocher and ochery earths, sienna and sienna earths, and umber and umber earths	5 per cent ad valorem.
1922	75	Ochers, siennas, and umbers, crude or not ground	1/8 cent per pound
		Washed or ground	3/8 cent per pound
1930	73	Ochers, siennas, and umbers, crude or not ground	1/8 cent per pound
		Washed or ground	3/8 cent per pound

Vandyke brown and other brown earth pigments, with the exception of ocher, umber, and sienna, have always come under the blanket clauses of the several tariff acts as a color or pigments not otherwise specially provided for. Since 1883 they have been dutiable at from 15 to 30 per cent ad valorem, the low duty under the act of 1913 and the high duty under the acts of 1890 and 1897. The present duty of 25 per cent ad valorem (par. 66, act of 1930) is the same as that under the act of 1922.

Table 11 shows the rates of duty under this classification which includes Vandyke brown and other earth pigments, with the exception as noted above.

Table 11.- Tariff rates on paints, etc., not elsewhere specified

Act of-	Par.	Tariff classification or description	Rates of duty, specific and ad valorem
1883	87	Colors and paints, whether dry or mixed, or ground with water or oil, and not specially enumerated or provided for in this act	25 per cent ad valorem
1890	61	All other paints and colors, whether dry or mixed, or ground in water or oil, not specially provided for in this act, and artists' colors of all kinds, in tubes or otherwise	Do
		All paints and colors, mixed or ground with water or solutions other than oil, and commercially known as artists' water-color paints	30 per cent ad valorem

Table 11.- Tariff rates on paints, etc., not elsewhere specified - Cont'd

Act of-	Par.	Tariff classification or description	Rates of duty, specific and ad valorem
1894	48	All other paints, colors, and pigments, whether dry or mixed, or ground in water or oil, or other solutions, including all colors in tubes, and not specially provided for in this act	25 per cent ad valorem
1897	58	All paints, colors, pigments whether crude or dry or mixed, or ground with water or oil or with solutions other than oil, not otherwise specially provided for in this act	30 per cent ad valorem
		All paints, colors, and pigments, commonly known as artists' paints or colors, whether in tubes, pans, cakes, or other forms	Do.
1909	56	All paints, colors, pigments, stains, whether crude or dry or mixed, or ground with water or oil or with solutions other than oil, not otherwise specially provided for in this section....	Do.
		All paints, colors, and pigments commonly known as artists' paints or colors, whether in tubes, pans, cakes, or other forms	Do.
1909	51	Enamel paints made with varnish	Do.
1913	63	Enamel paints, and all paints, colors, pigments, stains, whether crude, dry, mixed, or ground with water or oil or with solutions other than oil, not specially provided for in this section	15 per cent ad valorem
		All paints, colors, and pigments commonly known as artists' paints or colors, whether in tubes, pans, cakes, or other forms	20 per cent ad valorem
1922	68	Pigments, colors, stains, and paints, including enamel paints, whether dry, mixed, or ground in or mixed with water, oil or solutions other than oil, not specially provided for ...	25 per cent ad valorem
1930	66	Pigments, colors, stains, and paints, including enamel paints, whether dry, mixed, or ground in or mixed with water oil or solutions other than oil, not specially provided for ...	25 per cent ad valorem

EXPORTS

Statistics of exports are lacking. Although large quantities of mixed paints (containing in the aggregate a considerable quantity of these pigments) are exported, the exports of umber, sienna, and other brown earth pigments, either dry or ground in oil or water are probably negligible.

THE INDUSTRY IN FOREIGN COUNTRIES

Umber, sienna, and other brown earth pigments are found throughout the world, but certain countries are noted for their pigments because of their high quality.

Specially fine qualities of umber are produced in Cyprus, Sicily, Asia Minor, and the Netherlands. German deposits also have yielded considerable quantities, and at one time its umber ranked with that produced in Cyprus. Turkish umber, produced chiefly in Cyprus, is the world standard on this pigment and dominates the markets of all countries, although the production in Sicily is almost as large and of good quality.

Italian sienna likewise enjoys a world-wide reputation, the output coming primarily from Tuscany. The Harz Mountains in Germany are the only other important source of supply, although small amounts are produced elsewhere, notably in Sicily. The Sicilian product, however, is shipped to the mainland of Italy for further treatment.

Italy.- Table 12 shows the production of all mineral pigments in Italy for the years 1920 to 1929.

The chief producing centers in recent years are in the provinces of Verona, Cagliari, and Grosseto. Prior to the World War, the provinces of Siena and Grosseto were much more important producers, but immediately after the war and for some three or four years these provinces produced only minor quantities. During the past three years, however, Grosseto has increased its production until it has become one of the three leading producers. Other provinces which have produced in recent years are, Cuneo, Trento, Livorno, Udine, and several others of minor importance.

Leghorn for many years has been the marketing center for Italian siennas, and much of the crude earth is prepared there. Considerable umber from Cypress is also brought to this port for preparation and marketing.

Table 12.- Production of mineral pigments in Italy[1]
(Metric tons)

Province	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929
Verona	5,400	6,000	4,245	5,830	4,818	5,270	7,620	4,850	4,520	6,700
Cagliari	2,338	2,338	1,460	1,500	2,160	1,944	2,114	120	280	187
Vicenza	1,350	550	450	550	- -	- -	- -	40	- -	46
Cuneo	1,100	650	200	120	350	100	150	- -	- -	- -
Trento	1,000	30	30	90	64	300	146	40	70	70
Perugia	550	150	- -	330	- -	- -	- -	- -	- -	400
Siena	50	100	100	120	120	120	890	200	200	200
Grosseto	50	50	50	900	1,900	1,900	1,900	1,400	1,527	2,020
Livorno	- -	- -	- -	- -	- -	310	310	- -	143	45
Napoli	- -	- -	- -	- -	- -	- -	- -	100	120	100
Viterbo	- -	- -	- -	- -	- -	- -	- -	- -	- -	900
Friuli	- -	- -	- -	870	- -	119	229	- -	- -	- -
Udine	- -	- -	- -	- -	- -	- -	- -	311	200	- -
Roma	- -	- -	- -	- -	- -	- -	- -	- -	120	- -
Total	11,838	9,868	6,505	11,310	9,412	10,063	13,359	7,061	8,260	10,668

1/ Rivista del Servizio Minerario, Rome (Annual).

Cyprus.- Umber, known as Turkish umber, is produced chiefly in the Larnaka district of the island of Cyprus. At present there are two plants producing umber for market, their combined capacity being estimated at about 10,000 tons annually.

Practically all of the output from Cyprus is exported. The exports average about 5,000 to 6,000 tons annually. Prior to the World War, England and Italy were the principal destinations but to-day the United States purchases more than half of the total, with England and Italy following in the order named.

It has been estimated that Cyprus could produce 50,000 tons annually if the market could absorb such an amount.

Table 13 shows the exports of umber (Terra Umbra) for Cyprus for the years 1924 to 1929, inclusive. All of these shipments are made through the port of Larnaka.

Table 13.- Exports of terra umbra from Cyprus, 1924-1929[1]
(Long tons)

Country	1924	1925	1926	1927	1928	1929
United Kingdom	1,326	1,803	1,109	1,428	1,867	1,894
France	50	51	45	82	198	161
Germany	147	129	- -	1	3	5
Greece	- -	7	1	3	- -	5
Holland	- -	81	59	244	104	25
Italy	655	524	1,043	589	1,043	947
United States	2,569	3,372	3,104	3,196	2,575	3,550
Other Countries	84	2	- -	23	2	- -
Total	4,831	5,969	5,361	5,566	5,792	6,587

1/ Cyprus Year Book, Nicosia, Cyprus (Annual).

MARKETING AND PRICES

Domestic production of umber and sienna has never been large, and the output is generally controlled by a few producers who handle a wide variety of natural mineral pigments, both domestic and imported material, and have built up reputations for uniformity and high grade of products. The larger consumers of pigments prefer to deal only with such companies rather than buy each color from a different small producer. In this manner, imported umber and sienna and other brown earth pigments are sold side by side with domestic pigments, and due to the high standard of the imported materials, American pigments encounter severe competition. Some of the smaller producers market their product through brokers or jobbers.

The yearly price range for the various grades of umber and sienna both domestic and foreign, is shown in Tables 14 and 15. As will be seen, the prices quoted for foreign umber and sienna are only slightly higher than those for domestic pigments, despite their generally superior reputation.

Table 16 shows the yearly price range for Vandyke or Cassel brown.

Table 14.- Prices of umber, domestic and imported, in New York[1]
(Cents per pound)

Year	American, per pound, in carload lots		Turkish, per pound				
			Dry				In oil
	Burnt and powdered	Raw	Burnt and powdered	Burnt lump selected[2]	Raw and powdered	Raw in lumps	Burnt or raw
1915	2 -2½	2¼ -3	3 - 4	3 - 5	2 3/8-3½	3 - 5	11 - 14
1916	2 -2½	2 -3	3 - 5	3½ - 6	3 -3½	3½ - 5	11 - 21
1917	2 -5	2 -5	4 - 7½	4½ - 6	3 -6	3½ - 5	15 - 21
1918	3½ -4	3 -3½	4½ - 7	4½ - 6	4½ -6½	4 - 5	17 - 27
1919	3½ -4	3 -3½	5 - 7	5 - 7	[3]	[3]	25 - 30
1920	3½ -6	3 -6	5 - 7	5 - 7	4½ -6	6	28 - 30
1921	4½ -5	3½ -5	5½ - 7	-	6	-	24 - 28
1922	3¾ -4 5/8	3¾ -4 5/8	4¼	-	4¼	-	19 - 22
1923	3¾ -4 5/8	3¾ -4 5/8	4	-	4	-	19 - 22
1924	3½ -4½	3½ -4½	4	-	4	-	-
1925	3¼ -4½	-	4 - 6	-	4	-	-
1926	3 -3¼	-	4	-	4	-	-
1927	3	-	4	-	4	-	-
1928	2 7/8-3½	2 7/8-3½	4 - 6	-	4 -6	-	-
1929	2 7/8-3	-	4	-	-	-	-
1930	2 7/8	-	4	-	-	-	-

1/ Oil, Paint, and Drug Reporter, New York (Weekly).
2/ In ton lots.
3/ Nominal.

Table 15.- Prices of sienna, domestic and imported, in New York[1]
(Cents per pound)

Year	Italian Sienna		American Sienna
	In oil	Burnt	Burnt and powdered
1914	12 - 15	4 - 7	2¼ - 3
1915	12 - 15	4 - 7	2¼ - 3
1916	12 - 22	5 - 8	2 - 3
1917	16 - 32	5½ - 10	2 - 3
1918	18 - 28	6 - 15	2 - 4
1919	26 - 32	7 - 15	2½ - 4
1920	30 - 32	5½ - 16	2½ - 7½
1921	26 - 30	6½ - 15	4¼ - 7½
1922	21 - 25	6 - 14½	3¾
1923	21 - 25	6 - 14½	3¾
1924	4½ - 5½	-	3½ - 3¾
1925	3 - 4½	-	3
1926	3 - 4	-	3
1927	4 - 5	-	3
1928	5 - 5½	5 - 12½	3 - 4
1929	5½	-	3
1930	5½	-	3

1/ Oil, Paint, and Drug Reporter, NewYork (Weekly).

Table 16.- Prices of Vandyke or Cassel brown, domestic, in New York[1]
(Cents per pound)

Year	Dry	In oil
1914	2½ - 3	11 - 14
1915	2½ - 3	11 - 14
1916	3 - 4	11 - 14
1917	10	25 - 30
1918	4 - 7	25 - 30
1919	2½ - 4	25 - 35
1920	2½ - 5	(Not quoted)
1921	3½ - 5	25 - 28
1922	3½ - 4½	30 - 32
1923	2/ 4 7/8	30 - 32
1924	2/ 4 - 4½	30 - 32
1925	2/ 4 - 4½	-
1926	2/ 4 - 4½	-
1927	2/ 4	-
1928	2/ 4 - 4½	-
1929	2/ 4½ - 4½	-
1930	2/ 4½ - 4½	-

1/ Oil, Paint and Drug Reporter, New York (Weekly).
2/ Imported material.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

HOW AND WHY FATALITIES OCCURRED IN PENNSYLVANIA BITUMINOUS COAL MINES DURING THE FIVE-YEAR PERIOD, 1926 - 1930[1]

By W. J. Fene[2]

PURPOSE OF REPORT

A comprehensive study of all bituminous coal-mine fatalities occurring in Pennsylvania during the 5-year period, 1926-1930, was made to try to determine the factors that influence accidents which cause these fatalities. During this study some two thousand reports of fatalities were reviewed, and pertinent information as to causes and influencing factors were tabulated therefrom. These tabulations are discussed in detail, and in some instances recommendations are made with a view to reducing the hazards in certain classes of these accidents.

ACKNOWLEDGMENT

This study was made possible through the cooperation of Walter H. Glasgow, secretary of mines, Pennsylvania Department of Mines, who made available its records and extended every assistance and courtesy during this study.

SCOPE OF STUDY

The fatality reports studied cover the years 1926 to 1930, inclusive, and include 1,982 fatalities from all causes. This study covers all fatalities occurring in and around the mines, including those in mines employing fewer than ten men; this class is not required by law to report accidents, and the reports that are obtained are those that come to the attention of the district inspectors.

Nonfatal accidents have not been considered in this study other than to enumerate the number occurring in connection with compensation awards.

Fatal accidents have been classified as to causes, place in mine where accident occurred, nationality of those killed, age, occupation, years of experience, and time of day.

The factors which influence conditions that are likely to bring about fatalities from falls of roof and coal and also economic losses due to fatalities and injuries are classified and discussed.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6505."

2 Associate mining engineer, United States Bureau of Mines Safety Station, Pittsburgh, Pa.

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PENNSYLVANIA FATALITY RATES

The fatality rates for the bituminous-coal mines of Pennsylvania are lower than the average for the United States, although the natural conditions in many of the mines are as unfavorable as are likely to be found elsewhere. In the Pittsburgh coal bed, from which the greater part of the State's production is obtained, a treacherous draw slate occurs over the coal and is responsible for most of the fatalities from falls of roof.

In the compilation of the statistical tables used in this report, data were collected from the records of the Pennsylvania Department of Mines and from United States Bureau of Mines publications. In many cases the records of the two sources do not agree, chiefly because the Bureau of Mines records include those fatalities that occur in small mines which do not come under the State regulations and are not included in the records of the State Department of Mines.

In considering the fatality rates, consideration must be given to numerous factors, such as time involved, number of men in the mine, and production. Unquestionably, the fairest method of comparison of accidents is based on the fatalities of other accidents per million man-hours worked, which gives the rate per time of exposure in and around the mines. However, men work a varying number of hours in the mines, and it is difficult to procure exact figures as to the number of hours worked -- in fact, exact figures on man-hours of exposure are very seldom available in mining. The most readily available basis for figuring accidents on an approximate time of exposure is of 300 working days. Another basis for comparison of accidents is that of production per accident; when considered alone, this is merely a cost figure, as many factors enter into large or small production of coal per man. In mines that use the same mining methods and work the same coal bed under similar natural conditions, a comparison of rates on a tonnage basis would have a definite relative meaning; otherwise a comparison on this basis may mean little, or be unfair.

Table 1 is a comparison of Pennsylvania fatality rates with those of the United States as a whole for a period of 20 years. It is noted that the average yearly number killed in Pennsylvania bituminous-coal mines is 421, and the average of the United States is 2,344.

In this 20-year period, Pennsylvania produced 25.7 per cent of the bituminous coal in the United States, while the State was responsible for 18 per cent of the fatalities which occurred during this period.

There was a marked reduction in the number of fatalities during 1930; there were 300 killed as against the average of 421 for the 20-year period. The small number of fatalities for 1930 is the best for the past 30 years, with the exception of 1921 when production was below normal. The fatalities during the years 1911, 1913, and 1928 were considerably above the average, due to disastrous explosions occurring during these years. There has not been a major explosion (one in which five or more were killed) in Pennsylvania bituminous coal mines from March, 1929, to July, 1931.

The death rate per million tons of bituminous coal mined in Pennsylvania has not exceeded that of the United States, except in 1928, in which year one explosion cost the lives of 194 men. The average deaths per million tons of coal mined for Pennsylvania for the 20-year period was 2.87, as compared to 4.09 for the United States. The average number of

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tons of coal per fatality of Pennsylvania was 356,403 tons compared with 247,497 tons per fatality for the United States for the 20-year period 1911 - 1930.

In comparing the fatality rate on the basis of a thousand 300-day workers, it is seen that Pennsylvania's rate for 1928 was higher than that for the United States. However, the State's average rate on this basis for the 20-year period is approximately 26 per cent less than that for the United States, being 3.30 for Pennsylvania as compared with 4.45 for the United States for a 19-year period, the data for 1930 not being available for the United States.

Table 1.- Comparison of Pennsylvania (bituminous mines) fatality rates with that of the United States as a whole, 1911 - 1930, inclusive[1]

Year	Number killed		Deaths per million tons		Tons per fatality		Deaths per 1000 300-day workers	
	Pa.	U. S.	Pa.	U. S.	Pa.	U. S.	Pa.	U. S.
1911	515	2,656	3.62	5.35	276,096	186,887	3.63	4.97
1912	446	2,419	2.77	4.53	360,606	220,945	2.91	4.46
1913	611	2,785	3.53	4.89	283,086	204,685	3.61	4.70
1914	413	2,454	2.83	4.78	353,231	209,261	2.95	4.66
1915	442	2,269	2.81	4.27	356,154	234,297	3.12	4.44
1916	436	2,226	2.58	3.77	387,899	265,094	2.90	3.93
1917	494	2,696	2.89	4.14	346,304	241,618	3.04	4.25
1918	494	2,580	2.79	3.80	358,739	262,873	3.03	3.94
1919	401	2,323	2.73	4.19	366,797	238,464	3.05	4.28
1920	424	2,272	2.54	3.45	393,700	289,729	2.90	3.78
1921	284	1,995	2.48	3.94	402,983	253,832	2.82	4.20
1922	424	1,984	3.91	4.16	255,488	240,399	4.24	4.90
1923	401	2,462	2.37	3.74	421,898	267,223	2.82	4.39
1924	345	2,402	2.68	4.20	373,191	237,974	3.30	4.80
1925	312	2,234	2.31	3.84	433,547	260,461	2.83	4.65
1926	426	2,518	2.81	3.83	356,212	261,241	3.46	4.50
1927	356	2,231	2.70	3.73	369,789	267,978	3.33	4.43
1928	513	2,176	3.95	3.78	253,111	264,749	5.12	4.64
1929	387	2,187	2.72	3.59	367,832	278,380	3.69	4.54
1930	300	2,014	2.43	3.79	411,393	263,869	3.28[2]	-
Total	8,424	46,833	57.45	81.77	7,128,056	4,949,959	66.03	84.46
Average	421	2,344	2.87	4.09	356,403	247,497	3.30	4.45

1 Data of Pennsylvania rates from records of State Department of Mines, Data of United States rates from U. S. Bureau of Mines bulletins.

2 Estimated.

CAUSES OF ACCIDENTS

Table 2 shows the number of fatalities by causes that have occurred during the 5-year period, 1926 - 1930. As is the case in the United States as a whole, the greatest percentage of fatalities was due to falls of roof and coal, 53.08 per cent of all fatalities being ascribed to this cause; gas and dust explosions were responsible for 18.06 per cent, and haulage for 15.49 per cent of those killed underground. During the 5-year period,

fatalities caused by mining machines showed a gradual increase, much of which may be charged to the increased use of these machines.

Table 2.- Number of fatalities by causes, Pennsylvania bituminous mines, 1926 - 1930[1]

Cause of accident	1926	1927	1928	1929	1930	Total	Per cent of total
Killed underground:							
Falls of roof and coal	241	215	170	226	200	1,052	53.08
Haulage	65	63	67	62	50	307	15.49
Gas and dust explosions	69	10	227	47	5	358	18.06
Explosives	9	12	4	9	8	42	2.12
Suffocation from mine gases	-	-	1	3	-	4	0.20
Electricity	8	15	12	7	8	50	2.52
Mining machines	4	7	7	8	11	37	1.87
Shaft accidents	3	-	4	3	2	12	0.60
Other causes	7	10	6	6	5	34	1.72
Total	406	332	498	371	289	1,896	95.66
Killed outside of mine:							
Haulage	8	12	7	10	6	43	2.17
Electricity	3	3	2	2	-	10	0.50
Machinery	9	1	1	1	1	13	0.66
Other causes	-	8	5	3	4	20	1.01
Total	20	24	15	16	11	86	4.34
Grand total	426	356	513	387	300	1,982	100.00

1 Data obtained from records of Pennsylvania Department of Mines.

OCCUPATIONS OF THOSE KILLED

Tagle 3 shows the occupations of those killed and the percentage of the number employed, excluding those killed by explosions. This table gives an indication of the relative hazards of the different occupations, both underground and on the surface.

It appears that rockmen and timbermen have by far the most hazardous occupations underground, 0.527 per cent of those thus employed being killed yearly. Of the 80 rockmen and timbermen killed during the 5-year period, 68 were killed by falls of roof and coal. As a rule, men employed in these occupations are those who have had considerable underground experience, and the large death rate in this line of mining activity might indicate that they are not properly supervised and disciplined, and that they may be careless in performing their duties.

Next to those engaged as rockmen and timbermen, machinemen and helpers suffer a greater loss than any other class of underground workers. About 58 per cent of the machinemen killed during the 5-year period met death by falls of roof and coal, and 11 per cent in connection with transportation of the machines. The high percentage killed by falls of roof and coal may have been due to at least four factors: (1) In many cases it was necessary to

remove a post to make room for the machine; (2) the noise made by the machine did not permit the operators to hear the "working" of the roof; (3) the vibration of the machine on the coal and on the roof loosened the material to the extent that it fell; (4) the machine took so much space in its manipulation that enough props could not be placed.

Haulage employees rank third in death rate. Of the group engaged in haulage, the occupation of motorman's assistant, sometimes called brakeman or trip rider, is the most hazardous.

The death rate of underground officials is fairly large; it is difficult to explain the reason for this, as mine officials are supposed to be men who have had considerable experience and who know the dangerous conditions when they are encountered and how to handle these dangers safely. Of the 28 officials killed during the 5-year period, 12 were killed by falls of roof or coal, and 8 by haulage. Six of those killed by haulage were fire bosses, indicating a very bad practice in which fire bosses use locomotives during their inspections. The percentage of officials killed yearly on the surface is by far greater than in any other outside occupation. It is universally agreed that the idea of safety and the prevention of accidents should emanate from the officials; if this is true, then it appears from this record that officials are setting a bad example in not using the proper precautions to protect themselves.

While more machine and pick miners are killed than workers in any other occupation, the number employed is far greater than in any other occupation. The percentage of machine miners killed yearly compared with the total engaged in that occupation during the 5-year period was 0.228, and of pick miners, 0.218, from which figures it would appear that the relative hazard of the two methods of mining is slightly in favor of the pick miners.

Tagle 3.- Occupations of those killed and per cent of number employed, 1926 - 1930, Pennsylvania bituminous mines
(Exclusive of those killed by explosions)

Occupation	1926	1927	1928	1929	1930	Total	Employed yearly, average	Killed yearly, per cent
Underground								
Rockman and timberman	13	18	9	19	21	80	3,033	0.527
Machineman and helper	14	26	18	22	28	108	6,616	0.326
Motorman	8	11	8	6	15	48	3,376	0.284
Motorman's assistant	11	12	13	15	10	61	2,576	0.474
Driver	16	16	7	12	12	63	4,666	0.270
Car handler	3	3	5	-	1	12	889	0.270
Rope rider	3	-	1	-	2	6	856	0.140
Total haulage								0.307
Machine miner	141	147	118	137	122	665	58,353	0.228
Pick miner	94	65	66	85	50	360	33,067	0.218
Cager	1	-	2	-	1	4)		
Driller	2	-	-	2	-	4)		
Laborer	5	6	8	4	3	26)		
Machinist	1	2	2	-	1	6)	5,415	0.196
Mason	1	1	-	-	1	3)		
Roadman	2	-	2	1	-	5)		
Rock-duster	1	-	-	-	1	2)		
Machine loader	-	-	-	-	3	3)		
Mine foreman	2	1	1	2	1	7	1,471	0.095
Assistant mine foreman	4	1	1	2	2	10	1,197	0.167
Fire goss	2	3	2	2	2	11	856	0.257
Total officials								0.159
Electrician	1	1	1	4	1	8	1,035	0.154
Shot firer	3	1	1	1	1	7	956	0.146
Trackman	7	6	8	4	5	30	4,509	0.133
Pumper and pipeman	1	2	-	4	3	10	1,644	0.122
Bratticeman	1	-	-	2	-	3	810	0.074
On Surface								
Superintendent and foreman	3	1	2	1	1	8	477	0.335
Brakeman	1	-	-	-	-	1)		
Car trimmer	1	1	-	-	-	2)		
Laborer	5	2	5	-	1	13)		
Electrician	1	2	-	-	-	3)	11,664	0.103
Locomotive engineer	1	-	-	-	-	1)		
Motorman	1	2	1	1	-	5)		
Miner	1	-	-	2	1	4)		
Slate picker	3	1	2	-	-	6)		
Others	-	10	-	11	4	25)		
Coke employee	1	2	2	-	-	5	2,869	0.035
Carpenter	1	-	-	1	-	2	1,406	0.028
Tippleman	1	3	1	-	2	7	5,905	0.024
Total	357	346	286	340	295	1,624	153,646	0.211

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AGE AND EXPERIENCE OF THOSE KILLED

During the study of the records of more than 1,600 individual fatalities in Pennsylvania during the past five years, the age and experience of each victim was tabulated, and it was found that the average age of those killed during this period was 39 years and the average experience in mines was 15.5 years. This would indicate that mine fatalities are not, in general, due to lack of experience on the part of the victim, but to carelessness, lack of supervision, discipline, or some other deficient essential.

Lack of experience and supervision was, no doubt, responsible for many of the fatalities. Seventy-nine, or nearly 5 per cent, of those killed were under the age of 20 years, and 58, or about $3\frac{1}{2}$ per cent, had less than one year's experience in mining.

Of those killed during the 5-year period, 338, or 20.8 per cent were over 50 years of age, a number of whom had as much as 50 years' experience. The high death rate of the older men may be accounted for by (1) lack of agility in escaping from dangerous situations; (2) defective vision, being unable to perceive dangerous conditions in time to avoid them; (3) defective hearing, unable to hear warning given by working roof; and (4) obstinacy in not carrying out instructions, verbal or written.

There may be some excuse for deaths among inexperienced men, but the fact that the average experience of those killed was 15.5 years clearly indicates that incorrect methods of mining, training, or supervision, or possibly all of these, are being pursued, or lack of regard for danger and carelessness has been shown by the victims.

NATIONALITY OF THOSE KILLED

Table 4 shows the nationality of those killed during the 5-year period, 1926 to 1930, classified according to the average yearly number employed, number killed, and percentage killed. The experience during these five years indicates that the Irish and Horwats are the safest workers, with the Granish, Serbians, and Norwegians accounting for the highest average yearly percentage killed.

There are by far a larger number of Americans, including negroes, than of any other nationality employed in the bituminous-coal mines of Pennsylvania. No doubt a great many of those classified as Americans are men of foreign birth who have become naturalized citizens of the United States.

Four nationalities, including American, Italian, Slavonian, and Polish, represent approximately 76 per cent of all those employed in the bituminous-coal mines of Pennsylvania.

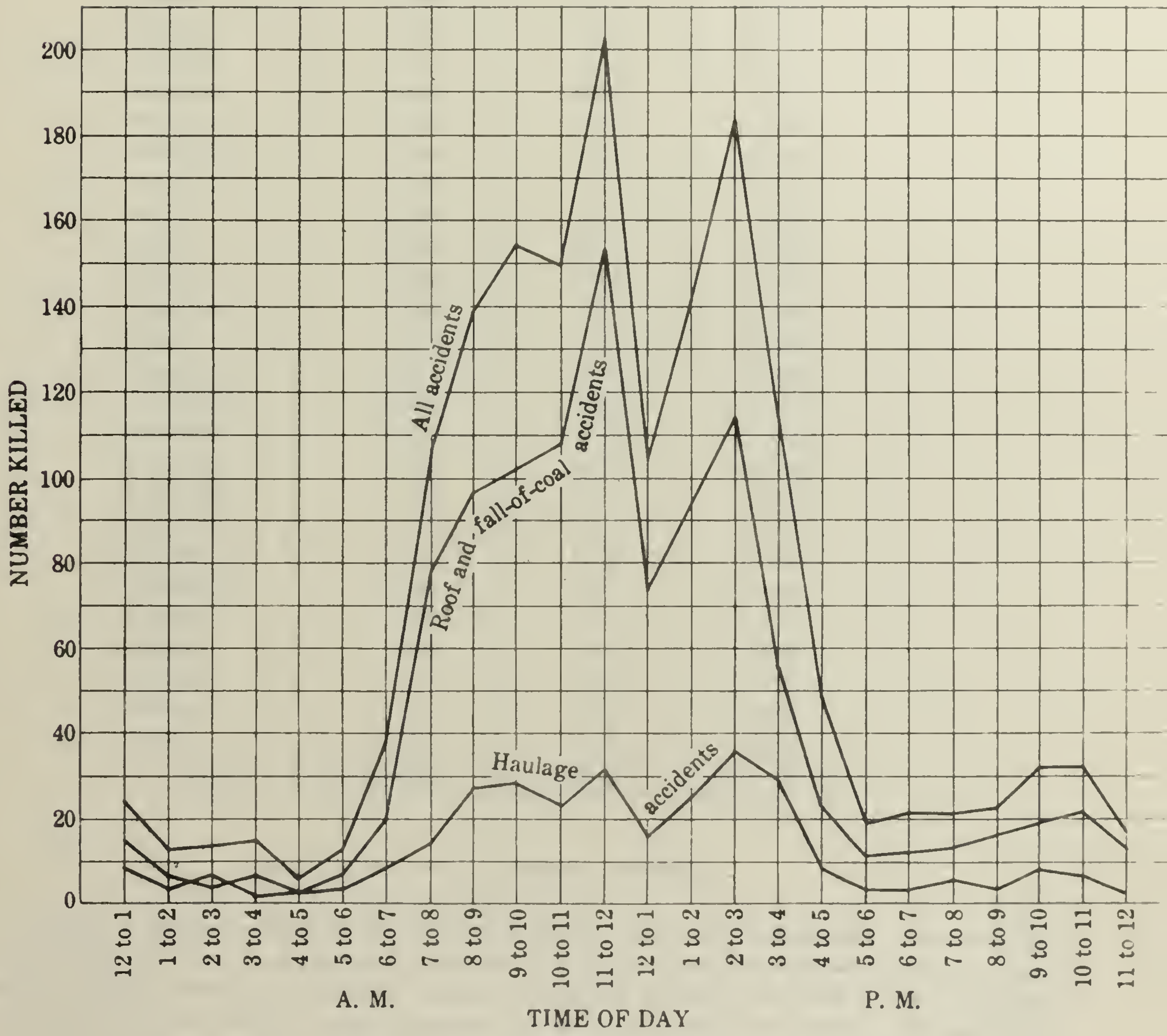

Figure 1.- Time of day when accidents occurred

Table 4.- Nationality of those killed during the period 1926 - 1930
(Exclusive of those killed by explosions)

Nationality	Average yearly Number employed	Average yearly Number killed	Percentage killed yearly
Irish	1,093	0.6	0.055
Horwat	1,843	1.6	0.087
Greek	364	0.6	0.165
Italian	14,856	25.4	0.171
Swedish	772	1.4	0.181
American	76,304	153.5	0.201
English	2,086	4.2	0.201
Hungarian	5,949	12.2	0.205
Belgian	463	1.0	0.216
German	2,022	4.4	0.217
Polish	12,425	29.0	0.233
Scotch	1,426	3.4	0.238
Croation	1,336	3.2	0.240
Welsh	476	1.2	0.252
Russian	4,304	11.0	0.256
Slavonian	13,376	35.6	0.266
Mexican	136	0.4	0.294
Austrian	6,229	20.4	0.328
Tyrolean	240	0.8	0.333
Rumanian	58	0.2	0.344
Finnish	159	0.6	0.377
Lithuanian	1,594	6.2	0.389
Spanish	195	1.4	0.718
Granish	216	2.0	0.926
Serbian	204	2.2	1.078
Norwegian	9	0.2	2.222
Others	5,511	3.2	0.058
Total	153,646	325.9	0.212

TIME OF DAY THAT ACCIDENTS OCCURRED

Figure 1 shows the time of day when accidents, excepting explosions, occurred during the 5-year period. It is noticeable that most of the accidents occurred during the forenoon, with the peak between 11 and 12 o'clock. There was a rapid decrease in the accidents between noon and 1 p.m., and a rapid increase up to 3 o'clock, after which the accidents occurred much less frequently. The peaks, of course, occur between 7 a.m. and 4 p.m., during which time the day shifts are in the mines. The high peaks occurring between 11 and 12 a.m. and between 2 and 3 p.m. are fatigue periods, and are also periods when employees are hurrying to complete particular jobs before lunch or to finish the day's work. The peaks here are due largely to falls of roof and coal accidents, a more detailed discussion of which is made later in this report. The rapid decline in the number of accidents occurring between noon and 1 p.m. may be accounted for by the fact that most of the underground and of the surface employees eat their lunch during this period.

CAUSES OF ACCIDENTS OCCURRING DURING 1926 to 1930

Roof and Coal Fall Accidents

During the 5-year period covered by this study, falls of roof and coal were responsible for 53.08 per cent of all fatalities in Pennsylvania's bituminous-coal mines, which is about equal to that of fatalities from this cause in the United States as a whole.

During the 5-year period, fatalities from this cause have decreased to some extent. During 1926 there were 241 fatalities and during 1930 there were 200; the record for 1928, the lowest of the 5-year period, was 170 killed. Falls of roof caused 887 fatalities in the 5-year period, and falls of coal caused 165.

In referring to Table 5, it is seen that 45.66 per cent of the roof-fall accidents, and 64.85 per cent of the coal-fall accidents occurred at the face of pillar workings. The high percentage of fatalities occurring at the face of pillar workings may be accounted for by the reasons that: (1) A great percentage of the coal mined is from pillar workings; (2) after removal of a portion of the pillar, the roof usually settles, making it more treacherous; (3) in many places timbers must be removed to effect a roof breakline, and many of those killed were in the act of removing timbers when they were caught by the roof; (4) in mechanical mining where pillar coal is loaded by scrapers or conveyors, timbers can not be set near enough to the coal to support the roof properly; and, (5) settling of the roof on the pillars usually crushes and loosens the coal, which may fall without warning.

Of the accidents occurring at the working face, 23.10 per cent of the roof falls and 23.03 per cent of the coal falls occurred at room working faces, and 12.40 per cent of the roof falls and 6.67 per cent of the coal falls occurred at entry working faces.

A comparatively large number of roof-fall accidents occurred on roadways or entries, which were largely due to horsebacks or slips that were not perceptible, and to posts being knocked out by derailed cars.

Table 5.- Location of falls of roof and coal accidents, 1926 to 1930

Place	1926	1927	1928	1929	1930	Total	Per cent of total	Per cent of falls
FALLS OF ROOF								
1. At working face								
(a) Pillar work	89	73	68	92	83	405	38.50	45.66
(b) Room	56	55	34	33	27	205	19.49	23.10
(c) Entry	25	26	13	30	16	110	10.46	12.40
2. In room	18	11	6	8	14	57	5.42	6.42
3. On roadway or entry	20	21	18	23	22	104	9.89	11.72
4. On slope	4	2	-	-	-	6	0.57	0.70
Total	212	188	139	186	162	887	84.33	100.00
FALLS OF COAL								
1. At working face								
(a) Pillar work	16	17	23	29	22	107	10.17	64.85
(b) Room	10	10	5	5	8	38	3.61	23.03
(c) Entry	1	-	2	4	4	11	1.04	6.67
2. On roadway or entry	2	-	1	2	4	9	0.85	5.45
Total	29	27	31	40	38	165	15.67	100.00
Grand total	241	215	170	226	200	1,052	100.00	100.00

Occupations of Those Killed by Roof and Coal Fall Accidents

Table 6 lists the occupations of those killed by roof and coal falls during the 5-year period, 1926 to 1930, inclusive. Of these occupations, most of those killed were employed as miners, mining machinemen, and timbermen. It is also noted that many drivers were killed during this period by falls, most of whom were killed when derailed cars knocked out props, allowing the roof to fall.

Of the miners killed, 61.66 per cent were machine miners, and 38.34 per cent were pick miners; the former represent 63.83 per cent and the latter represent 36.17 per cent of the average number of miners employed. From these figures it is seen that the pick miners, representing 36.17 per cent of all the miners, were responsible for 38.34 per cent of the fatalities from falls during the 5-year period. It would then appear that the hazard of falls in pick mining is slightly greater than where the coal is undercut by machines.

Table 6.- Occupations of those killed by roof and coal falls, 5-year period -- 1926-1930

Occupation	Number killed
Machine miners	492
Pick miners	306
Mining machinemen	63
Timbermen	52
Drivers	25
Trackmen	20
Laborers	16
Rockmen	16
Brakemen	9
Shot firers	7
Assistant foremen	6
Motormen	5
Pillar foremen	5
Wiremen	5
Fire bosses	4
Bratticemen	3
Conveyor loaders	3
Pumpers	3
Masons	3
Mine foremen	2
Roadmen	2
Driller	1
Machinist	1
Rope rider	1
Door boy	1
Signalmen	1
Total	1,052

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Time of Occurrence of Roof and Coal Fall Accidents

Referring to Figure 1, it is noted that comparatively few falls of roof and coal accidents occur between the hours of 12 midnight and 6 a.m., but between 6 and 7 o'clock in the morning these accidents begin to increase and reach a peak between 11 a.m. and 12 o'clock noon, with a decided drop between 12 noon and 1 p.m., from which time accidents begin to increase, reaching another high peak between 2 and 3 o'clock in the afternoon.

The high peaks occurring between 11 a.m. and 12 noon and between 2 and 3 p.m. may partly be explained by the fact that during these periods (1) the maximum area of roof is left unsupported, (2) unsupported draw slate falls before the miner chooses to take it down or support it with timber, (3) the miner experiences the maximum fatigue, (4) the workman is hurrying to complete his work before the lunch hour and before leaving the mine at the end of the shift, and (5) these periods immediately follow shooting in many mines.

FACTORS INFLUENCING ROOF AND COAL FALL ACCIDENTS

During the study of the fatality records of the Pennsylvania Department of Mines for the 5-year period, 1926 to 1930, all those factors which appeared to have an influence or bearing on the accident were noted and tabulated. The State inspectors investigate each fatal accident and prepare a complete report on it: Table 7 was compiled from these reports.

Table 7.- Factors influencing roof and coal fall accidents

	Number	Per cent of Total
Failure to test roof	137	8.71
Insufficient or improper timbering	450	28.61
Supervision and discipline	291	18.50
Post knocked out by car	42	2.67
Removing post	84	5.34
Horsebacks, slips (not perceptible)	171	10.87
Mechanical loading	13	0.83
Taking down slate	57	3.62
Blast fired between time of official inspection and accident	288	18.31
Mining machines	40	2.54
Total	1,573	100.00

During the period covered by Table 7, 1,052 fatalities occurred from falls of roof and coal. It will be noted, however, that the total number of cases listed in this table is 1,573, which may be accounted for by the fact that several factors may have influenced a single accident; for instance, insufficient or improper timbering may be due to supervision and discipline.

Failure to Test Roof

In 137 cases, or 8.71 per cent of the total, the accident may have been prevented had the victim tested the roof properly.

The general method of testing the roof is to "sound" it by striking it with a bar or pick.[3] The character of the sound is an approximate indication of the stability of the roof. If the sound is hollow or "drummy," the roof is probably loose and dangerous; it then should be taken down or supported with timbers. If a drummy sound is not noted this does not indicate that the roof is safe, for if a dangerous piece of roof is large it frequently does not give off hollow sounds which may be readily detected by the ear.

Inasmuch as the character of the roof can not always be determined by sounding, a more reliable method should be employed. The vibration method, which has been found to be more dependable than sounding and is readily applicable to coal beds less than 8 feet thick, consists of placing the bare fingers of one hand lightly against the roof, and with the free hand striking the roof sharply with an iron or steel bar. The degree of vibration of the roof against the fingers gives a fairly definite indication of the condition of the roof, provided that there is not too large a mass of loose material involved. When the roof is beyond the reach of the hand, a second rod may be held against the roof by the free hand while the roof is struck with a rod; and usually vibrations will be felt in the second rod if the roof is unsound or loose.

The vibration method of testing roof should be employed in addition to using sound indications, and rules should be adopted by all operating companies to provide that the roof and sides of working places shall be tested by the workmen when they go on shift, immediately after firing any shots, when a supervising official makes his visits, and at intervals not exceeding two hours during the shift.

The adoption of this rule, and its thorough enforcement, will be a large factor in preventing accidents from falls of roof and coal.

Insufficient or Improper Timbering

The greatest factor influencing roof and coal fall accidents evidently was failure to provide sufficient or proper timbering, since in about 28 per cent of the cases studied, the State inspector reported that the accidents might have been avoided if proper timbering had been employed. Many of these fatalities have occurred because the miner was unwilling to take the time to set supports or had delayed setting a post until he had performed some other task, and there is no question that in some cases the difficulty was due to lack of available timber.

So many fatalities occur from failure of one nature or another in connection with timbering that it is questionable whether it is in the interest of safety to rely wholly upon the miner to exercise his judgment as to the time and place of setting a prop, cross-bar, or other type of roof support.[4]

In many coal mines it has been found that little or no attention has been given to making or to following regulations for timbering; in others the matter has been fully considered.[5] Observations made at the latter mines have prompted the presentation of the following six essentials for mine-roof support in the interest of the former. In any scheme for protection against roof hazards there must be:

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3 Tomlinson, H., Reducing Accidents from Falls of Roof in Coal Mines. Part 3: I. C. 6225, Bureau of Mines, 1930, 11 pp.

4 Paul, J. W., Stop, Look, and Listen! The Roof is Going to Fall: I. C. 6032, Bureau of Mines, 1927, 3 pp.

5 Paul, J. W., Six Essentials for Mine-Roof Support. Part 1: I. C. 6225, 1930, 11 pp.

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[illegible]

[illegible]

[illegible]

1. Some definite system for setting timber; such a system is in the interest of economy in operation and of safety to the workman.

2. The adoption of diagrams for the use of underground officials and miners, which is a great aid in promoting regularity in setting timbers.

3. A definite system of diagrams illustrating the manner of placing timber, as the instruction of miners by underground officials is more easily accomplished and uniform practice is made possible.

4. Rigid inspection and close supervision; these measures are most essential in establishing methods which have to do with support of mine roof.

5. A record kept as to the manner in which timber regulations are executed or neglected; the use of timber or other forms of roof support, if not properly set, may be the cause of roof failure.

6. A plan or a scheme for the accomplishment of a definite course of action; there can be little or no discipline without a plan, and to enforce discipline there must be some penalty for neglect to follow instructions as set forth in the plan or scheme - in this case, systematic timbering.

When a plan of action has been adopted for roof support as a means of protection against accidents from falls of roof, it is recommended that a trained and experienced section or face boss be employed for each section of the mine to observe the roof constantly, to persist in pointing out roof dangers, to the miner in his working place, to give instructions as to how to safeguard the men from the danger, and to see that his instructions are promptly carried out.

Supervision and Discipline

The second largest factor that influences roof and coal fall accidents as revealed by this study is that of supervision and discipline.

Investigations of mine accidents by engineers of the United States Bureau of Mines indicate that the reduction of accidents is, other factors being equal, in direct proportion to the efficiency of the supervision, and those companies which have the best records are in general those that hold their foremen responsible for the safety of the men working under them and have a sufficient number of overseeing officials to allow of giving efficient supervision.

Reckless men are a reflection on the effectiveness of the management; either they should have never been employed or they should have been discharged before serious accidents occurred.[6] Careless men are careless chiefly because they are improperly supervised. The records show that miners with years of experience had neglected to support the roof when there was timber near by, and in many instances when they had been directed by the foreman to place the timber before doing any other work.

6 Miller, A. U., Safety as Affected by Supervision and Discipline: I. C. 6194, Bureau of Mines, 1929, 6 pages.

The strict and impartial enforcement of rules by the foreman is absolutely essential, and one of the best methods to insure proper supervision and strict enforcement of rules is for the management to hold the foreman directly responsible for the men under his supervision, but not to overload any one foreman with so much work of various kinds that he would be unable to give sufficient time to supervisory work.

It is believed that as a rule there is less supervision and discipline in mining than in any other important industry. In mining, 25 to 150 or more men are usually placed under one supervisory official whose section is often so scattered that he has to travel several miles to make one round. If the maximum number of men under one foreman were limited so that he could visit each working place at intervals of not more than two hours, he would then be able to instruct the men properly and see that his instructions were carried out.

Blasts Fired Between Official Inspection and Accident

In 288 of the 1,052 fatal accidents from falls of roof and coal, shots had been fired between the time of an official inspection and the accident. This would indicate (1) that shots are fired during the shift, either by shot firer or miner; (2) carelessness of the miner in not testing roof; (3) carelessness of the miner in not removing or properly supporting loose roof.

As a rule, shooting of the coal loosens the roof to some extent; this is especially true of the Pittsburgh coal bed, which is overlain by a draw slate that eventually has to be removed.

After shooting, the area of unsupported roof is likely to be large because the freshly shot-down coal prevents the placing of adequate supports. When loose roof is found in such places it should be removed before an attempt is made to load out the coal.

Most of the fatalities from falls of roof or coal after shooting are in those mines where the miner does his own shooting. In mines where the shooting is done by a shot firer, the place is usually inspected by him after the blasts; and if a dangerous condition exists, the miner is instructed how to handle it or is aided in the handling of it.

Fatalities occurring from falls of roof or coal after shooting could be greatly reduced if a longer waiting period was allowed before the men returned to the working face, during which period the material loosened by the shot or shots would fall without doing any damage. In mines where the shooting is done by shot firers after all other men are out of the mine, fatalities that occur from falls after shooting is, of course, unknown; and there is no question that this is much safer than any method of blasting with the shift in the mine. The general practice in Pennsylvania is to do the shooting during the working shift; and it is believed that many fatal accidents could be avoided if those shooting the shots were required to wait at least 30 minutes before returning to the working face, thus giving ample time for the smoke to clear away and for the preliminary falls to occur.

Falls of "Horsebacks" or "Pots"

Falls of "horsebacks" or "pots" whose presence was reported as not being perceptible, were responsible for 171 fatalities during the 5-year period. Generally, concealed horsebacks or pots are not detectable by ordinary testing, and in most instances they fall

without preliminary warning. Frequently, such accidents are classed as unavoidable because the pot or horseback that fell was not perceptible; but if systematic timbering is in effect and timbers set at certain fixed intervals, there is good reason to believe that many of this type of so-called unavoidable accidents would be avoided. On the other hand, it is generally the roof which appears to be good that falls and kills men; and here again, if systematic timbering were practiced, regardless of the apparent condition of the roof, a great many of these roof-fall accidents of all characters would be avoided.

Taking Down Slate

Fifty-seven of the fatalities due to falls of roof during the 5-year period occurred while the miner was taking down slate. Such accidents are frequently due to the carelessness of the victim in not properly protecting himself while performing this work, or in not using the proper tool. Pulling down loose slate or roof with a pick is dangerous work even though a man may be an experienced miner, and some accidents of this kind are more nearly unavoidable than almost any other kind of accidents which occur in mines. Long bars should be used for such work, and each miner should be instructed in the safe use thereof for releasing loose roof material.

Removing Post

There is little or no excuse for fatalities to occur from roof falls when posts are being removed; almost invaraiably the victim was either careless, lacked experience, or was not properly supervised. During the 5-year period covered by this study, 84 fatalities occurred by falls while posts were being removed. It is necessary in many cases to remove roof supports in pillar workings to effect a uniform break line or to secure effective caves or falls; such work should be done under the direct supervision of an official experienced in this kind of work. Posts or other timbers should not be knocked out with an ax or sledge, but post pullers should be used, with the operator standing well out of the danger zone.

Several fatal accidents occurred when posts were removed to make room for mining machines to operate along a face; when such procedure is necessary, safety posts should be set as near the face as possible before removing the post that is in the way of the machine. In some mines it is necessary to place a 3-piece timber set of two legs and a crossbar.

Post Knocked Out by Car

Most of the 42 men killed by falls when posts were knocked out by cars were haulage employees. Some of these fall accidents should rightfully be charged to haulage, as haulage equipment was directly responsible for them. The real remedy for prevention of fatalities from falls of roof caused knocking out of timbers by haulage equipment is the setting of timbers on haulage entries with legs of timber sets recessed into the rib, hence removed from effect of haulage wrecks or other haulage deficiencies. In a few cases, however, cars were pushed over the end of the track at working faces so that they knocked out posts; then the miner was responsible, as he should have provided blocks near the end of the rails.

Mining Machines

In 40 of the fatalities, mining machines were directly or indirectly responsible. In some of them it was necessary to remove posts to make room for the machine; in others

the coal gave way after the cut was made, and caught the operators; and in some cases timbers could not be set in sufficient numbers because of the room necessary to manipulate the machine.

Special attention should be given by mining-machine operators to the removal of props near the face to make space for cutting machines. If it is necessary to remove a prop, the roof should first be carefully tested, and if found to be unsound, a collar or crossbar should be placed to span the place where the prop stood before its removal; and before leaving the place, the prop should be replaced and the collar or crossbar also left in place. If, in moving a machine, it should knock down a prop, the prop should be immediately reset.

Mechanical Loading

In 13 cases, mechanical loading was given as a factor in influencing the fall accident; in some instances, it was necessary to leave a large area of unsupported roof to make space for the machine; in others, the noise made by the machine did not give the operators an opportunity to hear the warning sounds given off by the loosened roof. The big factor in roof falls in connection with mechanical loading of coal is the space occupied by the machine which requires the leaving of a large area of unsupported roof.

During 1928 (the last year for which these figures are available) approximately 2 per cent of the coal produced in Pennsylvania was loaded mechanically by 1.6 per cent of those employed in loading coal.

In mechanical loading, new conditions arise in roof support.[7] Where scraper loaders are in use there is usually a large area of roof of considerable length and varying width where props can not be placed because they would be in the way of the scraper. In some places, timber or steel in the form of a cantilever should be used, and where there is danger of the roof breaking the props, consideration should be given to the use of cribs which may be moved forward as the face advances.[8]

Shaker conveyors, flight conveyors, and belt conveyors usually admit of systematic placing of timbers near the face of the coal before it is shot down.

Loading machines require a certain amount of clearance for their proper operation, and all loading machines should be used only where roof conditions permit the taking of the necessary precautions against roof hazards. An adequate number of timbermen of experience and discretion should be kept employed to test the roof and place timbers to insure protection against insecure roof wherever mechanized loading practice is in effect.

All men employed in mechanized loading should be instructed how to test the roof by the vibration method and be required to make the test at frequent intervals, because the sound method of testing is of essentially no value in the presence of the noise of machinery.

7 Paul, J. W., What the Mine Foreman Can Do to Prevent Injury from Falls of Roof in Coal Mines: I. C. 6341, Bureau of Mines, 1930, 7 pp.

8 Harrington, D., Hazards in Connection with Concentrated Coal Mining: I. C. 6070, Bureau of Mines, May, 1928, 11 pp.

Regulations for Preventing Roof-Fall Accidents

In the preceding discussion it has been pointed out that roof and coal falls are the most common and the most serious of the ordinary causes of mine accidents; more than half of the yearly accidental deaths in the coal mines of Pennsylvania are due to this cause.

The killing of approximately 200 bituminous coal miners every year in Pennsylvania by falls of roof and coal constitutes one of the greatest problems in the mining industry of the State; it means the loss of about one life every mine working day. These accidents happen to experienced as well as to inexperienced men, and this accentuates the fact that something is inherently wrong and that measures to lessen accidents from roof and coal falls which have been taken in the past unquestionably are inadequate. The substance of provisions that have been adopted by some companies that have been very successful in reducing falls of roof accidents is given here:

1. It shall be the duty of all company officials to exercise continuous, unremitting efforts to prevent the occurrence of accidents from falls of roof and coal.

2. The strict observance and enforcement of all rules, regulations, and laws shall be a condition of employment for all officials and employees.

3. Disregard of rules, regulations, and laws with respect to roof support shall be cause of dismissal of any official or employee.

4. The official in charge of any section of a mine shall be held personally accountable for workmen in his charge who disregard the regulations as to care of roof and timbering.

5. A foreman shall not be placed in charge of a greater number of working places or men than he can visit with sufficient frequency during a shift to insure observance of the regulations.

6. Strict adherence to a definite system of timbering shall be compulsory. Additional timbers, necessitated by special conditions, shall be placed immediately, as determined by the supervising official or the worker.

7. Upon finding any portion of the roof in need of immediate attention, the supervising official shall remain and see that any dangerous material is either taken down or properly supported; or he shall order the workmen to vacate the place at once and shall display a sign of danger at the approach to the place until such time as the roof is made safe.

8. The miners shall have suitable tools for setting timber. The company should supply mechanical post-pullers and an adequate amount of suitable timber reasonably close to the point where it must be used.

9. Each accident due to a fall of roof or coal shall be thoroughly investigated by a committee of underground officials, in which the official in charge of the district where the accident occurred is not included. The latter shall be permitted, however, to submit a statement to the committee. A copy of the report of the committee shall be placed on the bulletin board for the information and benefit of all underground workmen.

10. In the interest of safety, miners should not be permitted to work alone. In general, two working places should be provided for each two miners, permitting them to work together in one place while the other is being cut.

HAULAGE ACCIDENTS

Haulage accidents caused 15.49 per cent of all fatalities during the 5-year period, which is nearly 20 per cent less than that for the United States as a whole. During 1930, there were 14 less fatalities from haulage than the average for the preceding four years.

Referring to Table 8, it is seen that the greatest cause of haulage accidents was getting caught between car or locomotive and rib. There are two principal reasons why such accidents occur, namely: (1) lack of clearance, and (2) derailed cars or motors due to defective track or equipment, or to defective practice or practices. The next greatest cause of haulage accidents was being run over by car or motor, which in many cases was due to the carelessness of the victim. The two causes - run over by car or locomotive and caught between car or locomotive and rib - are responsible for nearly 60 per cent of the haulage fatalities. Eighty-one per cent of the haulage accidents occurred in the haulageway, and 19 per cent occurred in working places.

Table 8.- Haulage accidents underground by causes, 1926-1930

Cause	1926	1927	1928	1929	1930	Total	Per cent
Switching and spragging	4	3	4	4	-	15	4.89
Coupling cars	-	-	-	1	3	4	1.30
Falling from trips	6	3	9	-	2	20	6.51
Run over by car or locomotive	14	19	15	23	7	78	25.41
Caught between car or locomotive and rib	21	23	24	19	19	106	34.53
Caught between car and roof	4	3	6	5	5	23	7.49
Runaway car or trip	5	7	7	7	11	37	12.05
Miscellaneous	11	5	2	3	3	24	7.82
Total	65	63	67	62	50	307	100.00

Occupation of Those Killed by Haulage Accidents

Those engaged in haulage are naturally more exposed to haulage hazards, and it would be expected that such employees would suffer a greater loss therefrom than those engaged in other occupations; however, experience in Pennsylvania during the 5-year period (refer to Table 9) shows that haulage employees suffered 43.32 per cent of the fatalities, while those engaged in other occupations suffered a loss of 56.68 per cent. Of those killed by haulage during the 5-year period, 40.39 per cent were employed as miners.

Of those employed in haulage, 1.07 per cent were killed in haulage accidents, while 0.135 per cent of those employed as miners were killed in haulage accidents.

Tagle 9.- Occupations of those killed by underground haulage accidents, 1926 to 1930, inclusive

Occupation	Number killed 1926	1927	1928	1929	1930	Total	
Haulage employees							
Motorman	8	8	8	5	8	37	
Motorman's assistant	9	9	11	11	9	49	
Driver	11	8	3	6	5	33	
Car handlers	3	3	2	–	3	11	
Rope rider	2	–	1	–	–	3	
Total haulage employees							133
Machine miner	14	23	16	23	15		91
Pick miner	7	5	8	7	6		33
Trackman	2	–	4	2	–		8
Fire boss	1	2	2	1	–		6
Timberman	1	–	1	2	1		5
Machineman helper	–	1	2	1	–		4
Machineman	1	1	2	–	–		4
Laborer	1	1	2	–	–		4
Electrician	1	–	1	1	1		4
Machinist	2	1	–	1	1		5
Pumper	–	1	1	–	–		2
Roadman	1	–	1	–	–		2
Mine foreman	–	–	1	1	–		2
Driller	1	–	–	1	–		2
Rockman	–	–	1	–	–		1
Pipeman	–	–	–	–	1		1
Total	65	63	67	62	50		307

Time of Day of Haulage Accidents

Referring to Figure 1, it is seen that the trend of the time of day that haulage accidents occur follows closely that of all accidents and of roof and coal fall accidents, the high peaks occurring between 11 a.m. and 12 noon and between 2 and 3 p.m. It is difficult to explain the reason for the peak between 11 a.m. and 12 o'clock, but probably fatigue enters into it. The principal factor causing the high peak between 2 and 3 o'clock in the afternoon is probably that of haste on the part of haulage and other employees in their endeavor to clean up for the day.

Conclusions as to Haulage Accidents

This study of haulage accidents points to the irresponsibility of the employee and to the fact that a large responsibility falls upon the employer. The former factor is a human element that may appear difficult to control; however, it is within power of the employer to correct this condition by intelligent selection of employees and by following up such selection with just treatment and with rigid discipline.

The prevention of haulage accidents depends largely upon proper equipment and proper maintenance of equipment, proper maintenance of haulageways, suitable layout of haulage for safe as well as efficient haulage of coal, and the adoption and enforcement of practices which will minimize haulage hazards, and one of these is care in the selection of the kind of men entrusted with haulage duties.

Fortunately, many accidents occur in haulage that do not result in injury to employees; such accidents almost invariably indicate that there were factors present which under similar circumstances might at another time contribute to the injury or death of an employee. All such accidents should be as carefully investigated as though they had resulted in an injury or a fatality, and the factors contributing to the accident should be corrected.

Haulage accidents generally entail greater economic loss than any other type of mine accident, excepting explosions and fires. As a rule there is an interruption of production; there is usually loss from damage to equipment; in some cases there is definite damage to the mine; and haulage accidents occasionally cause fires or explosions or both. The economic loss involved in haulage accidents should justify special effort on the part of the mine management to take all feasible precautions to prevent their occurrence.

As in all phases of accident prevention in mining, it is necessary to work out an adequate set of rules pertaining to safe and efficient transportation; and irrespective of the adequacy of the rules, results of the right nature will not be obtained unless the officials exercise strict discipline.

GAS AND COAL-DUST EXPLOSIONS

Pennsylvania's record with respect to gas and dust explosions was better during 1930 than for any year during three decades. No major explosions occurred during 1930, and only five persons were killed by explosions. The record of deaths from explosions was exceptionally high for 1928 due to one disaster in which 194 lives were lost. The decrease in number of lives lost due to explosions may be attributed to (1) more adequate ventilation of gassy mines; (2) the more widespread use of rock-dust, permissible electrical equipment, and closed lights; (3) the use of permissible explosives: and (4) the employment of State inspectors to examine underground electrical equipment. Most of the explosions in Pennsylvania bituminous coal mines in recent years have been caused by arcs from nonpermissible electrical equipment.

There is less excuse for the occurrence of mine explosions than for any other accidents in or around coal mines; if the known methods of the prevention of ignition, and propagation of gas and dust are employed, the chance of having explosions, or at least of having a widespread explosion, is so remote as to be almost non-existent.

ELECTRICAL ACCIDENTS

Electrical contact accidents were responsible for 50 deaths during the 5-year period, 1926 to 1930. Of these, 33 were caused by contact with trolley wires, 12 by contact with mining-machine feed wires, 1 by bar or tool striking the trolley wire, 1 by contact with mining machine, 1 by contact with haulage motor, and 2 were classed as miscellaneous.

Naturally, one would expect that those men having greater exposure to electrical hazards would be the persons who would suffer greater loss therefrom; however, the experience in Pennsylvania during the past five years does not bear out this fact. Of the 50 killed by contact with electricity, 34 were miners, 4 were mining-machine operators, 3 mining-machine scrapers, 2 motormen, 2 drivers, 1 rockman, 1 a pumper, 1 a trackman, 1 a timberman, and 1 a laborer.

As contact with trolley wires was responsible for 68 per cent of the deaths due to electrical contact, it is apparent that more precautions should be taken to prevent such contact. Where trolley wires are less than 6½ feet above the rail, they should be adequately guarded at all points, not only where men are required to pass under them, but throughout every foot of their length. The cost of universal guarding of trolley or other bare power wires which are less than 6½ feet above the floor is undoubtedly less in the long run than is the cost of the accidents resulting from failure to so guard the wires.

MINING-MACHINE ACCIDENTS

During the 5-year period, 1926 to 1930, 37 deaths were caused by mining machines. This is in addition to those killed by contact with mining machines and feed wires, and those killed by explosions caused by arcs from mining machines. Unquestionably, some of the 37 deaths caused by mining machines were due to carelessness on the part of the victim or the machine operator, but many were due to hazards brought about by the use of the machine or of loading or conveying machines used in the same places.

Of those killed by mining machines, 18 were machinemen's helpers, 13 were machinemen, 5 were miners, and 1 was a brakeman. There was little excuse for the deaths of the five miners and the brakeman, because there was no occasion for their being in proximity to mining machines while in operation; however, the fact that the mining machines in general use are poorly guarded or not guarded at all throws much of the blame for some of the accidents upon the mine operator or upon the manufacturer of the machines.

Ten of those killed were run over by mining machines, 8 were caught between the mining machine and rib, and 5 were caught in the bits of the machine; probably all of these deaths could have been avoided had proper precautions been taken.

EXPLOSIVES ACCIDENTS

During the 5-year period of this study, explosives accidents were responsible for 42 deaths, or 2.12 per cent of all deaths in Pennsylvania bituminous coal mines. Of those killed by explosives, 38 were miners, 2 were rockmen, 1 was a foreman, and 1 a loader.

Fifty-four per cent of all the explosives accidents occurring during this 5-year period were due to permature shots and delayed shots. Most of these deaths were due to the practice of blasting during the working shift. Some of the deaths from premature shots resulted from attempts to save a few cents in fuse and a little time; however, most of the deaths from this cause occurred where electric detonators were used. To avoid the possibility of premature shots, the uninsulated ends of the detonator lead wires should be twisted together or short-circuited by means of a metal clip, which should be maintained until ready to connect to the firing cable. In electrical blasting, a permissible portable battery should be used with at least 100 feet of cable.

Six men were killed by shots breaking through rib or pillar. Such accidents result from no warning, or lack of warning, from not guarding a place effectively, and from carelessness. These accidents illustrate lack of instruction or supervision and again are a direct result of blasting while the shift is working.

Probably most of these explosives accidents could have been avoided had qualified shot firers been employed. There are many advantages in the handling of all blasting operations by certified or qualified shot firers, who should examine the place for explosive gas and dangerous roof before and after shooting; if carefully chosen and supervised, they can see that the holes are properly placed, will use the proper amount of explosive, and will see that it is properly placed and tamped. They can be held responsivle for warning men in adjoining workings if blasting is done while men are in the mine, and in general can give a reasonable assurance that so dangerous a practice as blasting will always be done safely and efficiently.

SURFACE ACCIDENTS

During the past five years, 86 men were killed on the surface at bituminous coal mines in Pennsylvania, which was 4.34 per cent of all mine accidents occurring during this period. Fifty per cent of these surface accidents were caused by mine and railroad cars and locomotives, 15 per cent were caused by machiney, 12 per cent by electricity, and 25 per cent resulted from miscellaneous other causes, such as falls of men or material, or burns. Compared to the number employed, officials suffered a greater fatality rate than any of the other employees on the surface.

DEATH DUE TO NATURAL CAUSES

While deaths from "natural" causes are not considered mine accidents, it is interesting to note the number of such deaths occurring in the mines of Pennsylvania. During the 5-year period covered by this study, 77 men, or a yearly average of 15.4 died from natural causes; the average age was 47.4 years. In a majority of these cases the attending physician pronounced the cause of death to be heart trouble.

It is quite probable that the number of deaths from natural causes could be greatly reduced, or at least the span of life prolonged, if physical examinations were given to all applicants for work and reexamination of all employees required annually. These examinations should be complete and rigid, but they should not be used to discriminate in the selection of employees or to dismiss from service the less fit, but rather to acquaint employees with their physical condition, to give helpful advice, to give medical attention for any ailments discovered, and to try to allocate workers to activities within the scope of their physical ability.

RESPONSIBILITY FOR ACCIDENTS

Responsibility for fatal accidents is placed by the Pennsylvania Department of Mines in the report on each case; and during this study the responsibility for the accidents was noted, with the following result:

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

Table 10.- Responsibility for accidents according to Pennsylvania Department of Mines

	Victims
Responsibility in ordinary accidents:	
Victim	473
Official and victim	328
Accidental	291
Unavoidable	192
Officials	157
Equipment	141
Others	29
Undetermined	8
Responsibility for explosions:	
Equipment	253
Official	78
Victim	2

From this tabulation it is seen that the victim was adjudged responsible for 29 per cent of the individual accidents, and the victim and official were responsigle for 20 per cent. Most of the accidents charged to the victim and to the victim and official were caused by roof and coal falls and by haulage, where the victim had failed to test the roof, failed to set proper timbers, or failed to obey instructions; the last was certainly the result of improper supervision by the official.

It is probable that very few if any of the victims were ignorant of the necessity of using precautions to avoid the accidents that befell them, and it is difficult to understand why so many underground workers will work under conditions they know to be dangerous. Unless workers are compelled to take precautions, they will frequently take chances, and the only real remedy for this problem is intelligent supervision and strict enforcement of discipline.

Many of the accidents classed as accidental were haulage accidents in which apparently no one was to blame, and some were electrical contact accidents in which the victim could not be held directly responsible because of carelessness of for othere reasons. It is probable, however, that most of the accidents could have been avoided had the necessary precautions been taken or if the proper safeguarding of equipment had been in effect.

The accidents classed as unavoidable are mostly roof-fall accidents where horsebacks, pots, or slips were present and were not perceptible; in general many accidents of this type may be avoided if systematic timbering rules are in effect and timbers set at predetermined intervals, whether or not the roof appears to need the setting of timber to support it.

The accidents charged to equipment were mostly from haulage in which defective equipment was involved. As the operator is responsible for the upkeep of haulage equipment, he would likewise be responsible for fatalities resulting from such defective equipment. Equipment is also charged with being responsible for 75 per cent of the deaths due to explosions; the equipment in most cases was nonpermissible electrical machines, the arcs from which ignited gas or dust and caused explosions.

The question of placing responsibility for mine accidents is important in that it gives a basis upon which precautions may be taken to avoid similar accidents. However, regardless of responsibility for a fatal accident, the victim and the operator and the public all pay for it - the victim with his life and the loss of his earnings to his family, the operator by compensation and other costs, and the public in the increased cost of the coal it consumes as well as in numerous indirect costs due to the accident.

ECONOMIC LOSS OF ACCIDENTS

Frequently insufficient recognition is given to the economic or money loss involved in accidents by the company, by the industry, and by the public generally. If the same intelligent attention were given to accident prevention as is given to production and other phases of mining, the results would frequently mean the difference between profit and loss. It has been repeatedly demonstrated that the prevention of accidents pays big dividends in dollars and cents.

Statistics compiled by the Pennsylvania Bureau of Workmen's Compensation, for bituminous coal mining of the State, show that for the 5-year period, 1926 to 1930, there were 114,113 mine accidents (including all disabilities lasting two days or more), or that one man was injured for every 6.73 men employed.[9] (See Table 11.) The figures for 1930 show a decrease of 3,381 accidents reported over the average for the preceding four years.

Table 11.- Pennsylvania bituminous coal-mining accidents REPORTED to the Bureau of Wirkmen's Compensation[1]

Year	Total	Fatal	Nonfatal[2]
1926	24,677	443	24,234
1927	23,656	389	23,267
1928	22,123	543	22,580
1929	23,539	408	23,131
1930	20,118	337	19,781
Total ..	114,113	2,120	112,993

1 Data furnished by the Pennsylvania Bureau of Workmen's Compensation.
2 Includes all disagilities lasting two days or more.

Tagle 12 shows the total amount of compensation paid during the 5-year period to be $15,951,076; this is an average of $3,553 for each fatality, $1,322 for each permanent disability, and $85.67 for each temporary disability.

From Tables 11 and 12 it is seen that during the 5-year period there was a total of 114,113 fatal and nonfatal injury cases reported to the Bureau of Workmen's Compensation, while there was only a total of 65,872 cases in which awards of compensation were made. In other words, 48,241 cases were reported for which no compensation was awarded.

9 One authority shows this figure to be one man injured for every 3.36 men employed during the period 1925 to 1929.

Table 12.- Number of compensable accident cases and amounts of compensation AWARDED in the Bituminous-Coal Mining Industry of Pennsylvania[1]

Year	Total	Fatal	Permanent disability	Temporary disability	Total	Amount of Compensation: Fatal	Amount of Compensation: Permanent disability	Amount of Compensation: Temporary disability
1926	12,702	435	521	11,746	$ 2,958,443	$1,417,868	$ 651,040	$ 889,535
1927	12,393	423	492	11,478	2,796,944	1,344,944	566,642	885,117
1928	13,216	480	509	12,227	3,614,400	1,822,893	724,271	1,067,236
1929	14,025	403	482	13,140	3,392,061	1,498,953	642,816	1,250,292
1930	13,536	337	504	12,695	3,189,228	1,299,128	731,766	1,158,334
Total	65,872	2,078	2,508	61,286	15,951,076	$7,383,786	$3,316,535	$5,250,514

1 Data furnished by the Bureau of Workmen's Compensation of Pennsylvania. Includes accidents in mines employing fewer than 10 persons.

According to these figures, the average compensation cost per ton of coal produced in Pennsylvania during the past five years was 0.023 cent. It has been estimated that the medical cost, the wage loss, and the indirect cost is four times the compensation cost, which would mean a loss, due to accidents, of 0.115 cent per ton of coal produced if both the compensation or direct cost and the various indirect costs were included, or a loss of nearly 15 million dollars yearly to the bituminous coal industry of the State.[10]

Attention is called to the discrepancy between the fatality figures of the Bureau of Workmen's Compensation and those of the State Department of Mines. Those of the former include fatalities that occurred in mines employing less than 10 men, mines that do not come under the regulation of the State Department of Mines, and, therefore, are not included in its fatalities.

The economic losses due to accidents are not all covered by those than can be figured in dollars and cents; there is yet the social and community loss which can not readily be reckoned in money. During the 5-year period, 1926 to 1930, 4,840 widows and orphans were left dependent in Pennsylvania by fatal mine accidents. Although these dependents received a certain amount of compensation, in most cases such compensation is not adequate to provide the necessities of life, and the community, county, or State must contribute to the upkeep of these unfortunates.

CONCLUSIONS

The conclusions are given under each cause of accident, but two general statements may be made:

10 In a paper presented to the 44th Annual Meeting of the Coal Mining Institute of America, December 10, 1930, Rush N. Hosler, superintendent, coal mine section, Pennsylvania Compensation Rating and Inspection Bureau, estimated the direct cost of accidents in Pennsylvania bituminous coal mines to be 0.036 cent per ton and a total cost of 20 million dollars per year.

1. Although Pennsylvania's fatality record in its bituminous-coal mines has been consistently lower than that for the United States as a whole, there is still much room for improvement. This can come only through conscientious cooperation of the miner and the operator, the making of safety a condition of employment, strict observance of the State mining laws, and of up-to-date safety practices which are in advance of any laws.

2. The bituminous-coal mining industry is far behind other industries of the State in the promotion of safety; and the conditions in the coal industry are fast reaching the point where only those companies that eliminate unnecessary losses, such as the loss from accidents, will be able to survive under the intensive competition which now exists and which will probably become more rather than less intensive.

I. C. 6506

August, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

LOST-TIME ACCIDENTS IN SOME ALABAMA COAL AND IRON MINES DURING 1930

BY

F. E. Cash and H. B. Humphrey

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

LOST-TIME ACCIDENTS IN SOME ALABAMA COAL AND IRON MINES DURING 1930[1]

By F. E. Cash[2] and H. B. Humphrey[3]

The following information on accidents in Alabama during 1930 was compiled from data obtained from the companies represented in a course on accident prevention given by the United States Bureau of Mines in Birmingham. These mines produced nearly three-quarters of the tonnage of that State for the year, and included most of the larger operations in the district. The figures are probably representative of the mines which are doing recognized safety work, and are better than the average for all the mines in Alabama.

Table 1 lists the days lost in accordance with the weights given to fatalities and permanent disabilities, part and total, which is the scale used in the National Safety Competition for Mines, conducted by this bureau.

By way of explanation, temporary disabilities are weighted according to actual number of calendar days of disability, including Sundays and holidays -- that is, all days are counted except the day of accident and day of employee's ability to return to duty. Hernia is classed as a temporary disability to be charged with the actual number of calendar days during which the employee was unable to work.

Table 2 gives the number of accidents and the days lost from each cause at 45 coal mines and 7 iron ore mines, also the average frequency and severity rates.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6506."

2 - District engineer, U. S. Bureau of Mines Safety Station, Birmingham, Ala.

3 - Assistant engineer, U. S. Bureau of Mines Safety Station, Birmingham, Ala.

Table 1.- Scale of time losses for weighting deaths and permanent injuries to show severity of accidents

Nature of injury	Degree of disability in per cent of permanent total disability	Days lost
Death	100	6,000
Permanent total disability	100	6,000
Arm above elbow, dismemberment, or permanent disability of	75	4,500
Arm at or below elbow, dismemberment, or permanent disability of	60	3,600
Hane, dismemberment, or permanent disability of	50	3,000
Thumb, any permanent disability of	10	600
Any 1 finger, any permanent disability of	5	300
2 fingers, any permanent disability of	12½	750
3 fingers, any permanent disability of	20	1,200
4 fingers, any permanent disability of	30	1,800
Thumb and 1 finger, any permanent disability of	20	1,200
Thumb and 2 fingers, any permanent disability of	25	1,500
Thumb and 3 fingers, any permanent disability of	33 1/3	2,000
Thumb and 4 fingers, any permanent disability of	40	2,400
Leg above knee, dismemberment, or permanent disability of	75	4,500
Leg at or below knee, dismemberment, or permanent disability of	50	3,000
Foot, dismemberment, or permanent disability of	40	2,400
Great toe, or any two or more toes, any permanent disability of	5	300
1 toe, other than great toe, any permanent disability of	0	-
1 eye, loss of sight	30	1,800
Both eyes, loss of sight	100	6,000
1 ear, loss of hearing	10	600
Both ears, loss of hearing	50	3,000

Table 2.- Accident experience of some Alabama coal and iron ore mines in 1930

	Coal	Iron ore
Number of mines	45	7
Production, tons	10,781,718	2,354,251
Average number of men	15,893	1,828
Man-hours exposure	29,257,100	4,059,622
Accidents from falls	490	27
Days lost account of falls	177,497	31,525
Accidents from haulage	487	58
Days lost account of haulage	57,997	19,065
Accidents from electricity	41	1
Days lost account of electricity	60,457	13
Accidents from machinery	110	9
Days lost account of machinery	18,759	6,600
Accidents from explosives	17	4
Days lost account of explosives	13,057	12,103
Accidents from gas and dust	6	--
Days lost account of gas and dust	206	--
Accidents - miscellaneous	562	41
Days lost account of miscellaneous accidents	20,560	1,283
Total accidents	1,713	140
Total days lost	358,035	70,589
Frequency	58.5	34.50
Severity	12.22	15.70
Coal or ore per accident, tons	6,261	16,816
Coal or ore per day lost, tons	30	37
Coal or ore per fatality, tons	291,400	294,281
Fatalities	37	8
Lost-time accidents per fatality	46.3	17.5
Average days lost per accident including fatalities	203.5	504.2

By way of explanation, frequency rate means the number of all accidents (fatal, permanent, and temporary lost-time accidents) per million man-hours of exposure, and severity rate means the number of days of disability resulting from all accidents (fatal, permanent, and temporary lost-time accidents) per thousand man-hours of exposure.

DETAILS OF COAL MINES

The 45 coal mines represented in Table 2 included 1 mine producing less than 10,000 tons per year; 2 mines, 10,000 to 50,000 tons; 7 mines, 50,000 to 100,000 tons; 13 mines, 100,000 to 200,000 tons; 16 mines, 200,000 to 500,000 tons; and 6 mines, more than 500,000 tons.

Table 3 gives the classification with rates and accidents by causes for each class. Only one mine of under 10,000 tons reported working less than 60 days; the next group is insufficiently represented with only two mines, but the other groups include most of the principal producers of Alabama.

Table 3.- Accidents and rates by classes of coal mines

	Class I 1 mine under 10,000 tons	Class II 2 mines, 10,000 to 50,000 tons	Class III 7 mines, 50,000 to 100,000 tons	Class IV 13 mines, 100,000 to 200,000 tons	Class V. 16 mines, 200,000 to 500,000 tons	Class VI 6 mines, 500,000 tons and over	All classes, 45 mines
Coal, tons	1,600	80,300	516,200	1,633,300	4,420,800	4,129,600	10,781,800
Men	50	280	879	3,028	5,887	5,769	15,893
Man-hours	19,600	248,400	1,634,200	5,239,200	11,244,200	10,871,500	29,257,100
Falls	—	8	52	141	236	53	490
Haulage	—	4	41	119	203	120	487
Electricity	—	—	2	20	10	9	41
Machinery	—	1	14	30	39	26	110
Explosives	—	—	2	2	9	4	17
Gas and dust	—	—	—	—	1	5	6
Miscellaneous	—	11	53	153	166	179	562
Total accidents	—	24	164	465	664	396	1,713
Days lost	—	6,395	43,899	70,796	155,255	81,690	358,035
Frequency	—	92.6	100.6	88.7	59.3	36.4	58.5
Severity	—	26.00	26.91	13.51	13.90	7.51	12.22
Fatal	—	1	5	8	14	9	37

Falls.- Falls of rock and coal accounted for 490 of the 1,713 accidents, with 177,497 lost days. These were equivalent to 29 per cent of the accidents and 51 per cent of the days lost, as given in Table 4. Of the 37 fatal accidents, 20 were from this cause, showing that the danger from falls of rock and coal is rightly considered the most serious problem and that the fatal accidents from this source are high in frequency - 20 of 490, or almost 1 in 25.

In frequency, Classes II and III are both equally bad, as given in Table 5, with a rate of 32 accidents per million man-hours; and Class VI leads in safety with a frequency of only 5.

Falls of rock caused 1 fatality in Class II, 4 fatalities in Class III, none in Class IV, 9 in Class V, and 6 in Class VI; this leaves Class VI with the greatest severity rate, 6 fatals in 53 cases. This is an instance of mines with good safety practices and low frequency having much greater severity than more careless mines with greater frequency; the conclusion is that either accident frequency or accident severity alone is not indicative of the effort expended toward accident prevention, although the severity rate would probably show more definitely the comparative working hazard, natural or otherwise.

Table 4.- Accidents and days lost from different causes in coal mines

Cause	Accidents, per cent of total	Days lost, per cent of total	Fatal accidents
Falls of roof and coal	28.6	50.9	20
Haulage	28.4	16.6	4
Electricity	2.4	17.3	10
Machinery	6.4	5.3	1
Explosives	1.0	3.9	2
Gas and dust	0.4	0.1	-
Miscellaneous	32.8	5.9	-
Total	100.0	100.0	37

Table 5.- Accident frequency by causes in coal mines

Cause	Class II	III	IV	V	VI
Falls	32.2	31.9	27.0	21.1	4.9
Haulage	16.2	25.1	22.8	18.1	11.0
Electricity	0	1.2	3.8	0.9	0.8
Machinery	4.0	8.4	5.7	3.5	2.4
Explosives	0	1.2	0.4	0.8	0.4
Gas and dust	0	0	0	0.1	0.5
Miscellaneous	44.4	32.5	29.37	14.8	16.3

Haulage.- Almost the same number of haulage accidents were reported as falls, -- 487 or 28.4 per cent of the total number - but the time lost was much less than from falls, 57,997 days or 16.6 per cent of the total. Class III and IV mines had the greatest frequencies -- 25 and 23 accidents per million hours. Again Class VI has the lowest frequency, with 11. In all classes there were four fatal accidents in a total of 487, or 1 in 122 -- a good showing. Three of these fatalities were in Class V, giving it the greatest severity rating.

Electricity.- There were 41 electrical accidents equal to 2.4 per cent of accidents from all causes; but electrical accidents caused 17.3 per cent of the lost time, chiefly because of 10 fatalities. These would account for 60,000 of 60,457 days charged to this cause, leaving 31 accidents with only an average lost time of 14 days each. Seven of these fatal accidents were in Class IV, in mines having low, unguarded trolley wire and 13 lost-time electrical accidents. Class III had 2 accidents, 1 of which was fatal; Class V had 10, with 1 fatal, and Class IV had 9, with 1 fatal. The mines in Class IV are responsible for this unnecessarily bad record, chargeable almost wholly to unsafe methods and equipment.

Machinery.- Accidents from machinery were high, reflecting the increase in its use in the district. There were 110 accidents of this type, amounting to 6.4 per cent of all accidents. The time lost was equal to 18,759 days, or 5.3 per cent of the total. Of these 110 accidents only one was fatal, in Class VI. The others average 125 days each.

Explosives.- Explosives accidents were 17 in number, or 1 per cent of those reported, accounting for 13,057 lost days or 3.9 per cent of the total. There were two fatal cases, which is one in nine; subtracting 12,000 days for these from the total leaves only 1,057 days for the other 15 accidents, averaging 80 days a piece. Nine of these were in Class V, but Class III with two accidents had the greatest frequency of 1.2 per million man-hours.

Gas ignition.- Ignitions of gas caused only six accidents in these mines; this is equal to 0.4 per cent of the total and accounts for 206 lost days, equal to 0.1 per cent. There were no fatalities from this cause.

Miscellaneous.- Approximately one-third of the accidents (562 to be exact) are classed as miscellaneous, with 20,560 days lost time, or 6 per cent of the total days lost. The large number listed under this head is not because the companies did not classify them further, but largely because of different classifications given by different companies.

An average of 35 days lost time per accident indicates that most of these were relatively minor injuries.

General Remarks

As indicated by Table 3, Class II mines (50,000 to 100,000 tons) have the poorest safety record, with a frequency of 100.6 accidents per million hours of exposure and a severity of 26.91 days lost per thousand hours of exposure. This type of mine generally has older and poorer equipment, less adequate supervision, and often a short life in prospect.

Class IV mines (100,000 to 200,000 tons) have a severity rate (13.51) only half as great as that of Class III, and a frequency of 88.7, showing the saving effect of taking more precautions against accidents.

Class V (200,000 to 500,000 tons) has a severity rate (13.90) almost equal to Class IV, with a frequency of 59.3. This relatively high severity is due to nine fatal accidents by rock falls. Without the seven electrocutions in Class IV, mines of that size would have a fine record.

Class VI, comprising the largest mines with production in excess of 500,000 tons, leads in safety with a total severity rate of 7.51 days lost per thousand man-hours exposure, and a frequency of 36.4 accidents per million man-hours. The benefit of adequate supervision and a due regard for safety throughout the organization is apparent.

DETAILS OF IRON-ORE MINES

Less general information is available on the iron-ore mines. The seven mines included in Table 2 were large mines, producing 200,000 to 500,000 tons annually. It is probable that these mines represent closely the average conditions in the iron-ore mines of the Birmingham field. Comparison may be made with the general averages for the coal mines as given in Table 2.

Falls.- There were 27 accidents from falls of rock, or 20 per cent of 140 of all kinds (five of them fatal), or 1 in 5 compared to 1 in 25 for the coal mines. The days lost

from falls were 31,525 or 44 per cent of the total of 70,589. This greater danger in the ore mines is due to general greater height and a bad draw rock which breaks and falls behind the face, plus the fact that the greater specific gravity of the iron ore and its overlying strata causes more damage than the lighter coal-mine material.

Haulage.- Haulage accidents were more than twice the number of those from falls, amounting to 58 or 41 per cent, with one fatality. The days lcst from this cause were 19,065 or 27 per cent of the total. Steep grades, poor clearance, and running cars out of control contribute to this type of accident. Subtracting the lost time charged to one fatality, one permanent total disability, and one permanent partial disability, leaves 4,065 days to 55 cases, an average of 75 days each.

Electricity.- Electricity caused one accident of 13 days lost time.

Machinery.- Machinery was charged with nine accidents, or 7 per cent of the total; one was fatal. The lost time was 6,600 days, an average of 75 days for the nonfatal accidents. Accidents from scraper loading are involved in this class.

Explosives.- Explosives caused four accidents, two of them fatal, with a total lost time of 12,103 days.

Miscellaneous.- There were 41 miscellaneous accidents with 1,283 lost days.

COMPARISON OF COAL AND ORE MINES

The coal mines produced 6,261 tons per accident and the ore mines 16,816 tons - amounts about proportional to the specific gravity of the two products. The coal and ore per day lost were 30 tons for the former mines and 37 tons for the latter mines, and the quantity per fatality was practically the same at a little over 290,000 tons.

There were 46.3 lost-time accidents for each fatality in coal mines, and 17.5 in the ore mines. The total time lost per accident was 203.5 days for coal and 504.2 days for ore.

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I. C. 6507

SEPTEMBER, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

SAFETY INSPECTIONS IN AND AROUND IRON MINES IN THE LAKE SUPERIOR DISTRICT

BY

F. S. CRAWFORD

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY INSPECTIONS IN AND AROUND IRON MINES IN THE LAKE SUPERIOR DISTRICT[1]

By F. S. Crawford[2]

During the course of a study of the safety organizations of the various iron mining companies in the Lake Superior district considerable information regarding the methods of safety inspection in use was obtained which may be of value to other companies beginning safety work or contemplating changes in their methods.

A short description of the safety-inspection methods of eleven of these companies is given in this paper, and in the appendix are illustrated a number of forms used for safety inspections by several of the companies. Most of the companies whose methods are described use few forms for formal inspection. The one long form (10) attached which is used for a general inspection by the safety inspector appears to present a good method for enabling the inspector to check all branches of his inspection and to assure himself that everything has been covered.

All names of companies have been deleted from this paper and from the forms because the object is to disseminate information of constructive value rather than to point out the methods of particular companies by name.

SAFETY-INSPECTION METHODS

Company 1

Besides the regular inspections by the safety engineer, every supervisory official is charged with the safety inspections for all dangerous conditions which may come under his observation. This includes inspections by the mining captain, shift bosses, master mechanic, blacksmith, foremen, electrician, head pipeman, and surface bosses.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6507."

2 District engineer, U. S. Bureau of Mines Safety Station, Duluth, Minn.

Kinds of Inspections.- The Safety Engineer makes inspections underground for loose ground, badly placed timber, nails in timber or planks, headboards, ladder roads, covering of raises, staging, blasting lines, explosives, fuse, caps, "powder" houses, chute guards, trolley guards, safety zones, block-signal system, fire hazards, and cleanliness. On the surface he inspects for fire hazards, machinery guards, tracks, change houses, engine house, shops, test pits, and abandoned shafts, trestles, shaft houses, pulley stands, timber yards, and cleanliness.

The master mechanic, blacksmith foreman, electrician, head pipeman, and surface foreman make inspections of the equipment under their charge.

Frequency of Inspections.- Daily inspections are made by the captains and shift bosses. Weekly inspections are made by the safety engineer. Monthly inspections of blasting lines and fire extinguishers are made by the electrician. Bimonthly inspections of test pits and abandoned shafts are made by the safety engineer.

Following is a list of reports (forms 13 to 19 of which are appended to this report) customarily made out by the safety engineer and the chief electrician:

(13) Safety Report on Cages,
(14) Weekly Report of Lilly Hoist Controllers,
(15) Daily-Weekly Report on Skips, Cages, and Hoisting Cables,
(16) Weekly Report on Blasting Lines and Control,
(17) Method of Handling Timber at ----- Mine,
(18) Report on Safety Belts,
(19) Fire Extinguisher Inspection Report.

Company 2

Company 2 is small, and safety inspections are made by a representative committee from the foremen's conference and the works council. This is also part of the safety inspector's duties. There is no regular form of report, but the committee as a whole reports on matters which the members feel should have consideration.

Company 3

Company 3 is small and has no definite safety inspection, aside from that which is done by the shift bosses and mine foreman in their regular rounds. Other inspections are made by the county mine inspector, the State boiler inspector, and the State fire inspector; the latter two make annual visits, and the county mine inspector visits the operations approximately weekly.

Company 4

Company 4 has been comparatively small until recently, when their operations have expanded, and a safety engineer has been employed. The safety engineer makes daily inspections of all the plants operated by the company, and reports to the general superintendent by letter. No form for inspection reports has been used up to the present time.

Company 5

Company 5 works a small mine with three shafts. Two workmen spend one-half day on the safety inspection of each mine every three weeks just before the general safety meeting. They may be miners, trammers, timbermen, or other classes of workmen. The inspection is entirely informal, no report form being supplied for this purpose, and the report may be in the form of notes, r may even be verbal.

Company 6

Safety inspections are made by the range safety inspector, the underground safety inspector, the mining captains at each mine, the superintendents, assistant general superintendent, the general superintendent, and by every official.

These inspections are made every day. No formal report form is used, but a recommendation slip is handed to the mine superintendent and the foreman responsible. These recommendation slips are numbered and followed up very closely, one slip being made out for each item in question. They are carried as "unfinished" until they are completed to the satisfaction of the inspector. A report form is used only for the hoist inspection and the monthly summary of safety inspections.

Company 7

Company 7 is a comparatively large organization. The safety engineer inspects all properties regularly and continuously, taking three weeks to make the round of Michigan and Minnesota properties. Inspection is made of the surface and underground, in which particular attention is paid to the condition of guards for machinery, loose material, safe and unsafe practices, to seeing that safety regulations are being followed, that the hoisting cable is regularly inspected, that safety catches on the cages are tested every month, inspections are made for fire hazards, and to seeing that more than one exit is in good condition and that necessary precautions are taken to protect the workman.

A safety committee of workmen inspects each property each month, and makes a report at the general safety meeting. No special report forms are used. A weekly report by the safety engineer is sent to the general manager, assistant general manager, general superintendent, the claims manager, and the main office of the company.

Company 8

In another well-established organization, safety inspections of all properties and in every working place are made by the safety inspector and by committees of foremen and workmen. Some inspections are monthly, others quarterly, and others are annual.

Reproductions (1 to 10) of report forms are appended, and include the following:

(1) Cage Rider's Report - daily,
(2) Report of Inspection and Fire Equipment - quarterly,
(3) Report of Testing Cage Safety Catches - monthly,
(4) Report of Inspection of Skip and Cage Roads - weekly,
(5) Report of Inspection of Ladder Ways and Shaft Stations - weekly,
(6) Report of Testing Fire Doors - monthly,
(7) Report of Mine Rescue Apparatus - monthly,
(8) Report of Training with Mine Rescue Apparatus - monthly,
(9) Electrical Department Inspection Report of Equipment - quarterly,
(10) Mine Safety Inspection Report - monthly.

During 1929, there were 8,198 of these separate reports turned in to the safety department. The assistant to the safety inspector, who spends the major portion of his time training men in mine rescue and first aid, also spends part of his time checking these reports. He checks mainly as to recommendations which have been made. Since these reports go first to the superintendent of the mine, it is his duty to correct any difficulty at once. If the same difficulty is reported several times without action having been taken, it is a duty of the safety department to follow up the matter and see that the dangerous condition is corrected.

Company 9

Besides the inspection by the safety engineer, there are inspections by men in each department, such as the machinery mechanic, the head machinist, the mine electrician, the timber foreman, mining captain, shift bosses, and superintendents. There is also a periodical safety inspection at each mine by a committee of three, including a safety engineer, a superintendent, and the head mechanic from a mine in a different district. This committee is selected by the general manager for each inspection, which may take place semi-annually and sometimes more frequently, depending upon how much the mines are operating, but not made at any definite time. The superintendent and mechanic on the committee for an inspection of Michigan or Wisconsin mines of the company are selected from Minnesota, and conversely for inspection of Minnesota mines.

Inspections are made of the condition of wire rope and its fastenings; the condition of ore skips, man cages and fastenings, safety devices, such as cage safety dogs and guides, idler sheaves, pulleys, hoists, signal systems, bell codes, mine telephones; fire-fighting equipment; mine-rescue equipment; sanitation; boilers; electrical equipment; pumps; electric switches; shafts; manways including escapement shafts, raises, drifts; timbering; powder storage and handling; motor haulage; blasting; first-aid supplies; lighting; buildings and other structures; fencing footpaths, and other equipment.

Where conditions are continually changing, such as on equipment which is used for transporting the men into and from the mine, inspections are made every day. Other inspections are made weekly and monthly. A form of report is provided for inspection tests on such mechanical equipment as hoisting rope, hoists, rope fastenings, sheaves, idlers, skips, man cages, fire-fighting equipment, signal systems, telephones, handling of explosives, and a report of the cage rider. This form is used monthly, although the inspection is made ever day. The results of the other inspections are also reported monthly. Accident reports are made through a shift boss's report, report of the mine office, the doctor's report, the safety engineer's report, the State industrial commission report, and the report of investigation of the mine accident committee which consists of a mine captain, safety engineer, and an electrician or other mechanic selected by the superintendent, according to the nature of the accident and who function only in case of serious injury. The watchman also makes a daily report of safety conditions, and a form is provided for the report of the periodical safety inspection committee previously mentioned.

Company 10

Safety inspections are made by the safety engineer, assistant safety engineer, mining captains, shift bosses, old employees and new employees. Underground inspections relative to safety and cleanliness are made by the assistant safety engineer or the safety engineer once every month. All new employees act as safety inspectors for one day before going on their regular work; in this way all men underground will have acted as safety inspectors at some time.

Surface inspections relative to cleanliness are made by the engineer or assistant engineer every three months. Surface openings, such as test pits and shafts, are inspected once every month. All surface openings are covered where possible and fenced off. All fences and signs are inspected regularly every month, and a report is forwarded to all properties operated by the same company.

Four separate and distinct forms of safety inspections are made.

First: The foreman at the end of each shift makes out a safety and cleanliness report on every working place under his jurisdiction. The contents of this report are observed by the foreman on the opposite shift when he reports for duty, and the report is countersigned by him.

Second: The safety engineer submits to the safety committee a monthly inspection report covering every working place on the property, both on the surface and underground.

Third: New underground employees are assigned the duties of underground safety inspectors for the first day of their employment. They are accompanied through the mine either by the foreman or the mining captain and are required to make a written report covering their version of the safety conditions of the mine.

Fourth: Periodically, men are selected at random from the underground forces to act as safety inspectors. They are required to make a written report of their findings. This results in keeping the men always alert to dangers and has gradually instilled the safety idea into the entire crew.

No regular form is used for underground or surface inspections. The inspector writes up a report which is read and discussed at the safety meetings. The shift bosses make out a daily report (20) on safety and cleanliness underground which is read and discussed each morning at the mine.

Company 11

Safety inspections are made by the assistant safety engineers and the safety inspectors. Their inspection covers the condition and placing of guards and equipment, the cleanliness and sanitation of all openings in the mine, observation of the working habits of workmen, the handling of explosives, and other material. These inspections are made every day. The inspectors report verbally to the foreman as well as making a written report upon what they have found. A copy of this report goes to the foremen's committee, but no particular form of report is used for making this inspection. A daily rope inspection is made, results being entered upon the weekly head pipeman's report (11), which is sent to the master mechanic every Monday. This includes a daily inspection of the cage and safety "dogs," skips, sheaves, and counterweights. A drop test of the hoisting rope is made every month. Another report which is a statement of tests of each hoist made every day, and includes a test of the hoist controls, signals, brakes and controllers, is submitted monthly by the hoist inspector to the master mechanic's office on a form (12).

APPENDIX

(1) Cage Rider's Daily Report.
(2) Report of Inspection of Fire Equipment.
(3) Monthly Report of Testing Cage Safety Catches.
(4) Report of Inspection of Skip and Cage Roads.
(5) Weekly Report of Inspection of Ladder Ways and Shaft Stations.
(6) Monthly Report of Testing Fire Doors.
(7) Monthly Report of Mine-Rescue Apparatus.
(8) Monthly Report of Training with Mine-Rescue Apparatus.
(9) Quarterly Inspection Report - Electrical Department.
(10) Safety Inspection Report.
(11) Weekly Head Pipeman's Report.
(12) Hoist Inspector's Report.
(13) Safety Report on Cages.
(14) Weekly Report of Lilly Hoist Controllers.
(15) Daily-Weekly Report on Skips, Cages, and Hoisting Cables.
(16) Weekly Report on Blasting Lines and Control.
(17) Report Covering the Handling of Timber at the ----- Mine.
(18) Report on Safety Belts.
(19) Fire Extinguisher Inspection Report.
(20) Shift Bosses' Underground Safety Report.

10M 5-29 35436 S.P.Co. 1951

(806)

THE IRON CO.

MINING DEPARTMENT

Cage Rider's Daily Report ______________________ Mine

for ______________ Shift ______________ 19___

Day of Week Day of Month

Time on ______________ Time off ______________ Forenoon

Time on ______________ Time off ______________ Afternoon

1 Has anyone ridden on the cage without the cage rider?

2. Have the doors been shut when men are riding?

3. Is there to much crowding getting on and off the cage?

4. Have more men ridden on cage at any one time than the number designated by the superintendent?

5. Have any drills been carried to surface on the floor of cage with load of men?

6. At what hour was powder carried on cage into mine?

7. Have any men been carried on cage with powder?

8. Are signals in good condition?

9. Has the brakeman responded promptly and carefully to signals?

10. Remarks:

No

Mining Captain	Cage Rider

This report must be made out at end of shift.

Any violations of the above rules must be reported.

This report must be made in duplicate, one copy to be sent to the General Office and the other to the Captain to be kept on file at the Mine Office.

(914)

THE IRON COMPANY

MINING DEPARTMENT

REPORT OF INSPECTION OF FIRE EQUIPMENT

.... Mine

Made .... 192 ...

1. What is condition of hose? .... feet good .... feet poor
2. What pressure was used in test? ....
3. Is there a sufficient quantity of fire hose at mine? ....

   State number of feet ....
4. What is condition of hydrants? ....
5. Is the hydrant equipment ample? ....

   Remarks: ....

....

....

....

Inspector.

This report must be made out during the second week of January, April, July and October.

(2)

818

THE IRON CO.

MINING DEPARTMENT

Monthly Report of Testing Cage Safety Catches ____________________ Mine

____________________ 19____

5M 7-19 L. O. 2488

1. When were safety catches tested? ____________________
2. Did they work satisfactorily? ____________________
3. Do they now work satisfactorily? ____________________
4. Who were present when catches were tested? ____________________
5. Are other safety-catches in working order? ____________________

____

INSPECTOR

NOTE—Safety-catches must be tested in the presence of cage inspectur and the superintendent or the captain in the absence of superintendent.

(3)

911

____________________ 192__

THE IRON CO.

Report of Inspection of Skip and Cage Roads ____________________ Mine

2M 6-23. J.D.W. 3988

	CONDITION	REPAIRS NEEDED	REMARKS
________ Skip Road.	________	________	________
________ Skip Road.	________	________	________
Cage Road.	________	________	________
Detail of Repairs Needed	________	________	________

____

____

Noted ____________ 192__ Inspected by ____________________

____

NOTE:—Inspection must be made at least once each week.

(1)

(806)

The ______ Iron Company

Weekly Report of Inspection of Ladder Ways and Shaft Stations

.... Mine .... 19 ...

5M. 10-29. N.P.Co. J10579.

1. State condition of ladders.	
2. State condition of collars.	
3. State condition and location of any broken ladders, rungs or other dangerous places.	
4. State condition of shaft stations.	
5. Are entrances to shafts properly protected?	
6. Are lights at shaft stations in good condition?	
7. Are danger signs placed in ladderways above and below dangerous places until the same are repaired?	
8. REMARKS:	

Date ....

Noted by .... Inspector

The location and condition of any broken ladders or dangerous places in the ladderways must be reported at once to the Captain and the Inspector must make such temporary repairs as will leave things safe until regular repairs can be made. If such repairs cannot be made at once a danger sign shall be placed at entrance to ladderway and another at the bottom until repairs are made.

It shall be the duty of the Inspector to inspect all ladders and ladderways which are used as traveling roads, including the second outlet, once each week. It shall be his duty to report all changes or repairs which he has made and others which may be necessary. The entire mine is to be covered at one continuous inspection.

This report is to be made out in duplicate, one copy to be sent to General Office, the other to go to the Captain and be on file in the Mine Office. These reports are to go to the Captain and to the General Office on Monday of each week.

(825)

THE ______ IRON CO.

MINING DEPARTMENT

Monthly Report of Testing Fire Doors ________________ Mine

______________ 193___

1M-9-30 I.O. K2349

1. Date when doors were tested? ______________
2. Were all doors tested? ______________
3. Did they work satisfactorily? ______________
4. If not, what were the causes for failure? ______________

____

____

5. Are all doors in good condition? ______________
6. Is the connecting air line in good condition? ______________

____

INSPECTOR

NOTE—This report is to be made out in duplicate; one copy to be kept on file in the Mine Office and the other to be sent to the General Office.

The Iron Company

MINING DEPARTMENT (SAFETY)

Monthly Report of Mine Rescue Apparatus Station No.______

______________________, Mich.,______________192

HELMETS: Nos.						
Tubes and Connections						
Other Leather Parts						
Other Metal Parts						
BACK APPARATUS: Nos.						
Tubes and Connections						
Reducing Valves						
Breathing Bags						
Other Leather Parts						
OXYGEN STORAGE TANKS: Nos.						
Atmospheres						
ELECTRIC LAMPS: Nos.						
Date Battery Last Charged						
General Condition						
PUMP. Nos.						
State General Condition						
PULMOTOR: Nos.						
State General Condition						

Number of Fresh Potash Cartridges on hand .... 1 Hour .... 2 Hour ....

Number of Extra Oxygen Cylinders on hand (filled) ....

REMARKS: ....

....

....

.... Foreman

The Iron Company

MINING DEPARTMENT (Safety)

Monthly Report of Training with Mine Rescue Apparatus. Station No.________

________________, Mich., ________________ 19

TRAINING RECORD:

Name*	Mine	Date	Duration	Detail of Training	Condition of Wearer after Practice

REMARKS: ____

____

____

____

*Men undergoing training must first pass physician's examination and a certificate of the same must be kept on file at rescue station.

First Aid Supplies on Hand at Above Date

	Mine	On Hand	Wanted	Remarks
Stretchers				
Blankets				
Splints				
Bandage				
Cotton				
Gauze				
Spts. of Ammonia				

____

FOREMAN

THE IRON COMPANY

ELECTRICAL DEPARTMENT

QUARTERLY INSPECTION REPORT

SURFACE— Mine ....

Head-Frame, Landing and Tunnel—

Condition of Signals and Telephone ....

Condition of Lighting ....

Condition of Cables and Hangers ....

....

Are Armor and Sheath Properly Grounded? ....

UNDERGROUND—

Shaft Cables—Condition ....

When and by whom inspected through shaft ....

Levels	Haulage Track and Bonding	Trolley	Signals	Lighting

Main Pump House—

Cables .... Potheads ....

Switchboard .... Motors ....

Starting Apparatus ....

Are Motor Frames, Starters and Switchboard Frames Grounded? ....

Are Pyrene Extinguishers in Place? ....

....

Sump and Winze Pumps ....

....

Underground Hoists ....

....

Scrapers and Loaders ....

....

....

Ventilating Fans—A.C. or D.C. ....

Where Located ....

Condition of Motors ....

Condition of Starters and Switches ....

Are These Properly Protected and Fireproof? ....

....

Miscellaneous ....

....

Comments by Master Mechanic ....

....

Date .... Inspection by ....

Approved .... Chief Electrician .... C. M. E.

Noted .... Safety Inspector .... Gen. Supt.

.... Gen. Mgr.

This report is to be made quarterly in triplicate, two copies with the Chief Mechanical Engineer's comments to be sent promptly to the Central Office.

(9.

1M-6-27-I.O.-G-5125. (909)

THE IRON CO.

MINING DEPARTMENT

SAFETY INSPECTION REPORT

..................No..........

Mine ....

Date ....

THE IRON COMPANY

MINING DEPARTMENT

Safety Inspection Report

....No..........................

.... MINE DATE..

Surface—

1. What are conditions of roadways on Company's property over which workmen are obliged to travel to reach place of work?
2. Are such roadways properly lighted?
3. Are railway tracks used as roadways and cannot other ways be provided?
4. Are all open pits, caves, disused and abandoned shafts properly protected?

Change House—

1. Is dry lighted by kerosene lamps or electric lights?
2. Is dry sufficiently lighted?
3. Are accommodations ample for men employed at the mine?
4. What protection for safety from fire is provided?
5. Is fire protection in good order?
6. What are sanitary conditions?
7. What accommodations and appliances are provided for injured persons?
8. How many fire-fighting helmets are provided? And where kept?
9. Are the same kept in good order?
10. Are there men around the mine who understand the first care of injured men and the use of fire-fighting helmets?

Shafts—

1. Is protection at collar of shaft sufficient and in good order?
2. Is opening to shaft at timber tunnel properly protected and in good order?
3. Is protection at shaft stations sufficient and in good order?
4. Are there skip tenders and at what levels?
5. Is there a cage rider?
6. What tools are allowed with men riding in skip, cage or bucket?
7. Are projecting tools properly lashed to hoisting ropes?
8. How often are timbers in manways cleaned?
9. Are they in good condition?
10. Are working stations sufficiently lighted?
11. Are there passageways at all levels around hoisting compartment?

12. Are same in good condition?

13. Are there guides for the bucket?

14. What style of crosshead is used?

15. What clearance is provided on guides?

16. Is stopper securely fastened to hoisting rope at least seven feet above rim of bucket?

17. Are guides and crosshead kept in good condition?

18. Is there more than one outlet to surface?

19. Are there connections between levels other than the main shaft?

20. Is the second outlet kept in good condition?

21. Is condition of pipes, electric wires and conduits safe?

22. Are steam pipes covered or protected from accidental contact?

23. Is the general condition of the compartment through which men are hoisted safe for men as regards lagging, timber, guides, etc.?

Sinking—

(When more than one shaft is being sunk, separate report must be made for each.) The following questions apply to sinking new shafts and also deepening shafts:

1. In lowering is bucket stopped 15 feet above the bottom of shaft?
2. Is shaft suitably covered while sinking is going on?
3. Are only safety hooks used for bucket?
4. Are there ladders reaching to the bottom of the shaft while sinking?
5. State what kind of ladders and if in good condition?
6. Is bell rope within reach of men in bucket at bottom of shaft?

General Mining—

1. Is general condition of timbering or other means of support throughout the mine satisfactory?
2. Are mine maps clear and accurate for purpose of inspection?
3. Are all dangerous places fenced off?
4. Are proper danger signal boards displayed at all dangerous places?
5. Are candles or lamps left burning after shift?
6. Are sumps securely planked over?
7. In passageways, are roofs and walls securely lagged?
8. Are winzes and raises in direct line of drift?
9. Are winzes, raises and open stopes properly guarded?
10. Are ladderways in winzes and raises located in drifts properly protected by hatches?
11. Are all chutes in winzes and raises properly protected by gratings?
12. Is proper provision made for safety of men working at chutes?
13. Are the communications between contiguous mines in good condition?
14. What is the condition of ventilation in different parts of the mine?
15. Are sufficient dry closets maintained?
16. While the responsibility for the safety of the roof and walls in the individual places is upon the workman, are the same also inspected by superintendent, captain and shift bosses, and how often?
17. Are the rules for safety pillars on boundaries observed?
18. In approaching workings which are known to be flooded are proper precautions taken?
19. Complaints from workman—report if any?

Main and Sub-Levels—

1. Are levels and sub-levels properly ditched to prevent water from accumulating, making passage unsafe?
2. Are there any other conditions which might be dangerous for passageway for men?
3. Are tracks and switches in safe condition?
4. Is condition of wiring safe?
5. Is condition and arrangement of piping safe?
6. On haulage roads, are drifts properly lighted?
7. On all main drifts is there a manhole or refuge every 100 yards?
8. Are they kept clear?
9. Are the rules governing the employment of motor and brakeman carefully enforced?
10. Are the rules governing the operation of electric motors strictly enforced?

Explosives; Manner of Handling—

1. Are explosives stored on surface and are the company's rules strictly complied with?

2. Are caps stored on surface, and where?
3. How often are explosives taken into mine?
4. Are packages of explosives plainly marked, giving date and strength, also name and place of manufacture?
5. Is there more than 72 hours supply of explosives stored in one place in the mine?
6. Is the same stored 50 feet from any other supply of powder?
7. How is dynamite thawed? 8. Is the method employed safe?
9. Are conditions as to blasting complied with?
10. Are the rules posted in conspicuous places?
11. Do men in preparing dynamite for blasting always use skewers and cap crimpers?
12. Before blasting are the necessary warnings given?

Ladderways—

1. Do rungs of ladders exceed 12 inches part?
2. Are ladders placed 3 inches from wall of shaft or other openings?
3. Are there platforms for ladders at least every 24 feet?
4. Are there manholes for ladders?
5. Do ladders project at least 3 feet above sollars, or are hand rails provided?
6. Are any ladders inclined backwards from the vertical?
7. Are rungs made of wood and iron?
8. Is there a complete ladderway from the lowest workings to surface?
9. Is there any loose rock or timber left at the top of ladderways, where it might accidentally fall or be pushed into same?
10. Are ladders and ladderways kept in good condition?
11. How often inspected? 12. Are stairways kept in good condition?

MECHANICAL.

Hoisting –

1. Have printed rules, governing duties of hoisting been furnished to hoisting engineer and receipt taken for same?
2. Are the company's rules governing duties of hoisting engineers properly posted?
3. Are they maintained in good condition?
4. Is speed of hoisting and lowering men over 1,000 feet per minute?
5. How often are machinery and safety appliances inspected?
6. Are cages, skips or buckets used for hoisting men, inspected daily?
7. Are the cages properly enclosed and protected?
8. Are signals in good order; is there a duplicate system of signals?
9. Is signal code posted in engine house and at each level?
10. Is moving machinery properly guarded?
11. Is hoisting plant equipped with an overwinding device?
12. Is the same in good working order?
13. Are safety devices tested once each month by the superintendent or captain and inspector jointly?
14. Are hoisting ropes running through engine house protected for safety of men?
15. Are unnecessary persons excluded from the engine and boiler houses?

Hoisting Ropes—

1. Are ropes inspected every 24 hours? 2. What is the present condition of rope?
3. How often is section of rope tested?
4. When rope is lowered to bottom are there two full turns left on drum?
5. Is the further end of rope secured by six clamps or bolts?

Surface—

1. Are sheave stands and sheaves in shaft houses easily accessible and arranged for the safety of the men whose duties require them to oil and repair the same?
2. Is the machinery and belting in shops properly protected for the safety of employes and others?
3. Is the fire pump hydrant equipment provided for fire protection, ample?

4. Is the quantity of fire hose provided, ample? 5. When last inspected?
6. What is condition of same?
7. Where are lubricating oils and grease stored?
8. Is storage place kept in safe condition?
9. Where are kerosene and inflammable oils stored?
10. Is storage place kept in safe condition?
11. Are hoisting ropes, guy wires and piping on surface properly protected?
12. Is protection provided for safety of employes and others during construction and repair work?

Boilers—

1. Are safety devices provided for boilers kept in good working condition?
2. Are general conditions in boiler house safe?
3. How often are boilers inspected?
4. Are recommendations promptly complied with?
5. Are all steam pipes covered or protected from accidental contact?
6. Is boiler room properly lighted?

Engine House—

1. What is condition of piping and general equipment as regards safety?
2. Is engine room properly lighted?
3. Is basement of engine house used for storage of materials?
4. Is the same arranged for safety of men?

Compressors—

1. Is moving machinery properly guarded?
2. What is condition of piping around compressor as regards safety?

Electric Engines and Dynamos—

1. Is moving machinery properly guarded?
2. Is switchboard railed off to prevent handling by outsiders?
3. Are all wires carefully strung to prevent accidental contact?

Pump Stations—

1. Is moving machinery properly protected? 2. Are electric wires safe?
3. Is station properly lighted?

Top Tram—

1. Is moving machinery properly protected? 2. Is wiring properly protected?
3. Are counterweights safe?
4. Are walks and hand rails provided on permanent trestles?
5. Can the conditions for safety of men be improved?

Crusher—

1. Are belts and shafting properly protected?
2. Are proper railings provided for all walks and openings?
3. Are electric wires properly protected?
4. Are stairs and ladders in safe condition?
5. Are all oiling places safely accessible?

Steam Shovel—

1. Is all moving machinery properly protected?
2. Are all gears covered?
3. Is shovel in good condition for the work being done?
4. Are runways and stands protected by railing?
5. Is proper care taken to prevent trestle legs from falling and injuring workmen?

General Applications—

1. Are accidents to persons and equipment reported promptly?
2. Are occurrences of fire, floods, extraordinary caves, and similar accidents, dangerous to life or health, reported promptly.
3. Do superintendents, captains and shift bosses, instruct employes as to their responsibility?
4. Is discipline strict or lax?
5. Are persons allowed in or around the mine except by permission?

WEEKLY HEAD PIPEMAN'S REPORT

Week Beginning ______________

C A G E S	Monday	Tuesday	Wednesday	Thursday	Friday	Saturday
Condition of Rope Attachment						
Broken wires in rope						
General Condition						
Doors						
Gates						
Safety Dogs & Springs						
Wheels & Boxes						
Any broken wires at clips						
Miscellaneous						
Date of Change						

S K I P S	Monday	Tuesday	Wednesday	Thursday	Friday	Saturday
General Condition						
Bails						
Guides						
Wheels & Boxes						
Broken wires at clips						
Miscellaneous						
Date of Change						

S H E A V E S

General Remarks ______________

COUNTERWEIGHT

Condition ______________

Date of Pipe Lubrication ______________

REMARKS: ______________

HOIST INSPECTOR'S REPORT

______________ Hoist Month ______________ 19____

Date	Does Main Throttle close at proper time?	Does slow down device work?	Dist. from dump mark to trip point of overwind in inches.	Time in secs. to set brake with wedge in.	Time in secs. to set brake with wedge out.	Are gate or chair signals alright?	Time of Skip Inspection:	Time of Cage Safety Dog Inspection.	Is brake travel alright with cold brake?	Do slack rope signals work?	Is operating controller alright?	Is control panel alright?	Does signal bell work?	Does return bell work?	Signature.
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THE -------- MINING COMPANY

SAFETY REPORT ON CAGES

Date ______________

Date Tested	Name of Cage	Place Tested	By Whom	Cage Drop	Remarks
Mar. 25					

(13)

THE ------ MINING COMPANY

------, Michigan

WEEKLY REPORT OF LILLY HOIST CONTROLLERS

Date ______________

PLACE	CONDITION
No. 4 Skip Hoist, South Drum	
No. 4 Skip Hoist, North Drum	
No. 4 Cage Hoist, South Drum	
No. 4 Cage Hoist, North Drum	
No. 5 Cage Hoist.	

Remarks:

Chief Electrician

(14)

THE -------- MINING COMPANY
______, Michigan

Daily) REPORT ON SKIPS, CAGES AND HOISTING CABLES
WEEKLY)

PLACE	CONDITION	DATE
No. 4 Shaft Skips		
No. 4 Shaft Cages		
No. 4 Shaft Skip Rope Shackles		
No. 4 Cage Rope Shackles		
No. 4 Shaft Skip Ropes		
No. 4 Shaft Cage Ropes		
No. 5 Shaft Cages		
No. 5 Shaft Cage Rope Shackles		
No. 5 Shaft Cage Rope		
No. 4 Crusher		
No. 4 Skip Dump		

Foreman

Foreman

(15)

THE ______ MINING COMPANY

______, Michigan

WEEKLY REPORT ON BLASTING LINES AND CONTROL

NO. 4 SHAFT	CONDITION	DATE
24th Level		
25th Level		
26th Level		
27th Level		
28th Level		

Chief Electrician

(16)

THE ______ MINING COMPANY

______________ Mine

The following is a report covering the handling of timber at the ________ Mine:

23rd Level Foot Side:

No. 2 Raise - Timber handled with timber cradle. Timber hoist is at top of raise on the 6th sub.

Nos. 4, $5\frac{1}{2}$, 7, $8\frac{1}{2}$, 10 and 11-3/4 Raises - Timber handled with timber cradle, timber hoists at the bottom of raises.

23rd Level North Crosscut:

Nos. 21, 23, and $24\frac{1}{2}$ Raises - Timber handled with timber cradles. Timber hoists in 21 and 23 raises are on the 7th sub, and in $24\frac{1}{2}$ raise, the timber hoist is on the 10th sub.

24th Level No. 1 West Drift:

No. $25\frac{1}{2}$ Raise - Timber handled with timber cradle. We use two timber hoists in this raise, one hoist at the bottom of raise and one on the 7th sub.

Nos. 17, 20, $21\frac{1}{4}$, $22\frac{1}{2}$ and 24 Raises - Timber handled with timber cradle. Timber hoists at the bottom of raises.

25th Level No. 1 West Drift:

No. 3 Raise - Timber handled with timber cradle. Timber hoist at the bottom of raise.

We have a total of 17 timber cradles in use at the present time. Our practice is that wherever we are hoisting timber, we use a timber cradle. There is only one place in the mine where timber is being handled without a timber cradle. This is on the 23rd level north crosscut, No. 27 raise. The timber for this place is handled in $24\frac{1}{2}$ raise by timber cradle to the 8th sub, and carried over to the 27th raise, then hoisted with slusher hoist from the 8th to the top of the raise. There is only one gang of miners working in this place and it is not considered necessary to use a timber cradle.

All four of our timber hoists have been turned around so that the rope runs under the drum instead of over the drum, and they will be guarded as soon as possible.

____________, Safety Engineer.

(17)

REPORT ON SAFETY BELTS

April 2, 1930.

The following is a report on the results of the tests made on the safety belts. Sixteen belts were tested on the above date, and one out of this number failed to stand the test.

Belt No.	Size of Hemp Rope	Remarks
1	3/4"	O.K.
2	"	"
3	"	"
4	"	"
5	"	"
6	"	"
7	"	Belt Broke in Center.
8	"	"
9	"	"
10	"	"
11	"	"
12	"	"
13	"	"
14	"	"
15	"	"
16	"	"

Note: A new belt was put in place of Number 7, numbered the same.

LOCATION OF BELTS

Nos. 1 and 2	Blacksmith Shop
Nos. 3 and 4	Carpenter Shop
Nos. 5, 6, 7, 8	23rd Level No. Crosscut
Nos. 9, 10, 11, 12, 13	24th, 25th and 26th Levels
Nos. 14, 15, 16, 17	23rd Level Foot Side
Nos. 18, 19	In warehouse on stock

These belts were all tested on April 2, 1930, by dropping them six feet with a weight of 120 lbs. All belts were oiled before they were put back in service.

_______, Safety Engineer.

THE ______________ MINING COMPANY

______________ MINE

Shift Bosses Daily Underground Safety Report DATE ______________ SHIFT ______________

LEVEL	CONT. NO.	WORKING PLACE	CLEANLINESS	SAFETY	REMARKS

SHIFT BOSSES DAILY UNDERGROUND SAFETY REPORT

At the end of each shift, each boss is required to make out this report, showing the condition of each place over which he has charge. It must be signed by the boss making it out and also by the shift boss going on shift.

CONTENTS NOTED:

______________ BOSS IN CHARGE OF OPPOSITE SHIFT

______________ SHIFT BOSS IN CHARGE

THE ______ MINING COMPANY
FIRE EXTINGUISHER INSPECTION
March 21, 1930

	1 Qt. C. T.	2½ Gal S.A.	2½ Gal. F.T.	Remarks	Date Refilled	Date of Inspection
Doctor's office--near entrance	1			O.K.	7/19/29	3/17/30
Mine office--foot of stairs	1			"	11/15/29	3/17/30
Mine office--head of stairs	1			"	12/23/29	3/17/30
Laboratory--near entrance	1			"	11/15/29	3/17/30
#4 Captain's Dry	1			"	11/15/29	3/17/30
#4 Riverside Dry--near door			2	"	2/21/30	3/18/30
Warehouse--foot of stairs			1	"	6/14/29	3/17/30
Warehouse--near office	1			"	10/15/29	3/17/30
Carpenter shop--near tool room	1			"	5/17/29	3/17/30
Carpenter shop--near tool room			1	"	11/30/29	3/17/30
Electric shop--on door	1			"	11/15/29	3/17/30
Machine shop--near tool room	1			"	12/23/29	3/17/30
Blacksmith shop--near office	1			"	3/17/30	3/17/30
Watersupply--near door	2			"	9/18/20	3/18/30
#4 Engine House--near switchboard	2			"	9/19/28	3/17/30
#4 Engine House--center of east wall	2			"	8/20/29	3/17/30
#4 Engine Hoist--near ore hoist	2			"	6/14/29	3/17/30
#4 Engine House--center of basement	3			"	7/18/29	3/17/30
#4 Engine House--near cell of structure	2			"	11/15/29	3/17/30
----- Boiler Room--head of stairs	1			"	11/15/29	3/17/30
----- Engine House--on door	2			"	7/19/29	3/17/30
Club House--main stairway	1			"	7/19/29	3/17/30
Clubhouse--basement stairway	1			"	7/19/29	3/17/30
#4 Shaft House--motormen's room	1			"	11/29/29	3/18/30

	1 Qt. C.T.	2½ Gal S.A.	2½ Gal F.T.	Remarks	Refilled	Inspection
#4 Shaft House--near crusher control	1			O.K.	11/29/29	3/18/30
#2 Truck	1			"	1/27/30	3/18/30
4th Level #2 Shaft Pump Room	2			"	6/27/29	3/19/30
22nd Level Fan	2			"	1/23/30	3/19/30
22nd Level Powder House	2			Taken Out.		3/10/30
23rd Level Machine Shop	2			O.K.	2/22/30	3/19/30
23rd Level Pump House	1 Gal. C.T.			"	6/12/29	3/19/30
23rd Level Hoist Room	1			"	1/23/30	3/19/30
23rd Level Pump Room	2			"	10/16/29	3/19/30
23rd Level Fuse House	1			"	8/8/29	3/19/30
23rd Level Powder House	1			"	8/8/29	3/19/30
24th Level Sub Station	1			"	3/16/29	3/19/30
24th Level Powder House	1			"	6/15/29	3/19/30
Welding Torch and Tank Cart	1			"	2/21/30	3/19/30
#4 Engine House--center of east wall	1 Gal. C.T.			"	6/12/29	2/19/30

C.T. - Carbon Tetrachloride
S.A. - Soda Acid
F.T. - Foamite

I. C. 6508 September, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MILLING PRACTICE OF THE KIRKLAND LAKE GOLD MINES (LTD.), KIRKLAND LAKE, ONTARIO

BY

JOHN DIXON

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING PRACTICE OF THE KIRKLAND LAKE GOLD MINES (LTD.), KIRKLAND LAKE, ONTARIO[1]

By John Dixon[2]

INTRODUCTION AND ACKNOWLEDGMENT

This paper is one of a series on milling methods and costs being published by the Bureau of Mines.

Acknowledgment is made to J. B. Tyrrell, managing director, and Wm. Sixt, manager of the Kirkland Lake Gold Mines, for permission to publish this article.

The Kirkland Lake Gold Mining Co.'s mill is located on Government Road at the west end of the productive part of the Kirkland Lake mining area, 1 mile west of the village of Kirkland Lake. The crushing plant is connected to the shaft house of No. 2 shaft, which is now the only operating shaft.

The mill, which stands about 60 feet from the shaft house and midway between it and the Government Road, to the south, has an average capacity of 160 tons per day and consists of one unit. Grinding is done in cyanide solution by an 8-foot by 30-inch Hardinge ball mill and a 5½-inch by 16-foot tube mill. The cyanide plant employs counter current decantation in Dorr thickeners, followed by filtration on an Oliver filter.

Water for milling and tailings disposal is mine drainage supplemented by water purchased from the municipal plant of the village of Kirkland Lake. Hydroelectric power is purchased from the Northern Ontario Light and Power Co., a subsidiary of the Canada Northern Power Co.

ORE TREATED

The ore at present carries about $13 in gold. A little silver is present and is paid for by the mint, but is of no commercial importance. The ore is altered and silicified country rock. The country rocks in the mine are lamprophyre, quartz porphyry, syenite, and diabase, and any of these rocks may be altered and impregnated with gold up to ore grade.

The gold is very finely disseminated and some is probably in the form of gold telluride, although this has never been definitely proved. Lead telluride has been recognized, and tellurium can always be found in the furnace products from the refinery. Gold-bearing pyrite accounts for some of the gold. Values are not entirely unlocked for cyanidation or concentration at 300 mesh, although that seems about the economic limit of grinding at present.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6503."

2 - One of the consulting engineers, U. S. Bureau of Mines, and mill superintendent, Kirkland Lake Gold Mines, Ltd., Kirkland Lake, Ontario, Canada.

Gangue minerals are acid and basic feldspars, augite, hornblende, carbonates, and quartz. Physically the ore is generally tough, hard, and close-grained, without cleavage planes or other lines of weakness.

Mill feed carries 2 per cent of moisture.

There is no change, physical or chemical, in the broken ore lying in the stopes.

HISTORY OF PLANT OPERATIONS

The mill started operating in April, 1919, as a counter-current decantation plant treating $5 or $6 ore. It was closed during the summer of 1919, owing to a miners' strike. It was reopened in November, 1919, and operated continuously until January, 1924. During this period no radical changes or additions were made in the mill. The grade of ore varied from $6 to $10 per ton, and extraction was from 85 to 90 per cent. Grinding was about 70 to 85 per cent minus 200 mesh. The mill was closed from January, 1924, till October, 1926, due to the exhaustion of known ore bodies. Milling was resumed in October, 1926. As ore from lower levels was treated, the grade became higher and the gold more difficult to dissolve. Finer and finer grinding became necessary to maintain the percentage of extraction, and the absolute value of the tailings went up with grade of ore.

In 1927, 1½-inch cast-iron balls were substituted for flint pebbles in the tube mill. The change was made without trouble by feeding the metal instead of flint and using a mixture while the flint charge was grinding out. The tube mill is now operating with a low ball charge without trouble, and its capacity could be increased from 50 to 75 per cent by increasing the charge of balls.

In February, 1930, an 8 by 8-foot classifying cone was put in closed circuit with No. 1 thickener, No. 1 agitator, and the tube mill. In July, 1930, this was replaced by a 16 by 4-foot Dorr thickener running as a classifier, as shown in the flow sheet (fig. 1). The underflow from this machine is pumped by a 4-inch diaphragm slime pump directly to the feed box of the tube mill, and the overflow goes to the secondary agitation and then to the decantation system.

With finer grinding, colloids began to give trouble by trapping high-grade solution in floccules of coagulated slime in such a way that it could not be separated by decantation. This problem was met in 1928 by installing an Oliver filter to be used after the last step of decantation, as shown in the flow sheet.

In April, 1927, Crowe vacuum apparatus was installed. This resulted in saving about two-thirds of the zinc-dust formerly required for precipitation, as well as giving smoother plant operation and higher-grade precipitates.

The various changes mentioned have resulted in slightly raising the percentage of extraction without increase of costs, in spite of higher grade and more refractory ore.

METHOD OF MILLING

Figure 1 is the flow sheet of the mill. Figure 2 is a diagram showing the end elevation of the mill. The flow sheet shows roughly, in plan, the course of the ore through

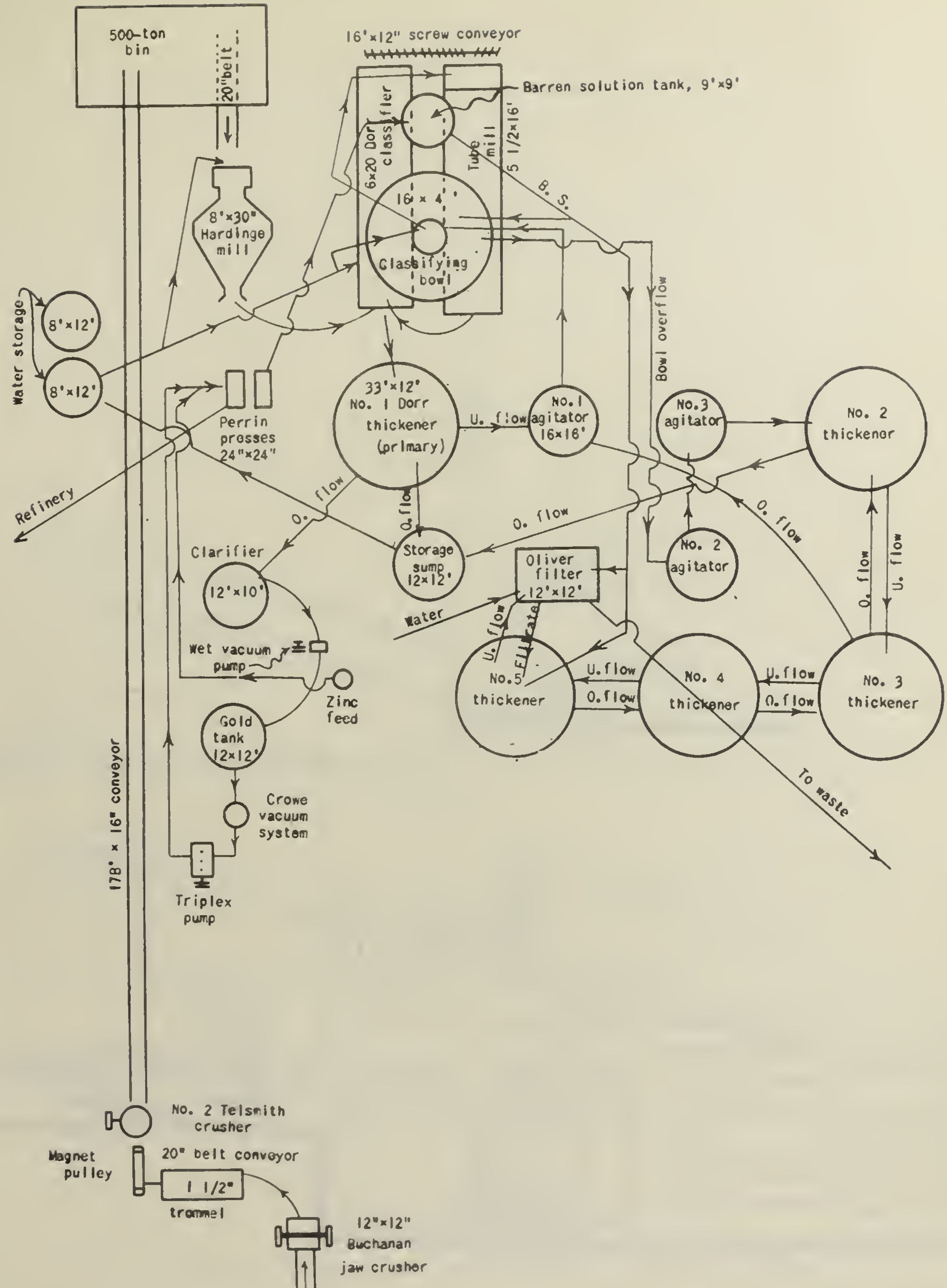

Figure 1.- Mill flow sheet

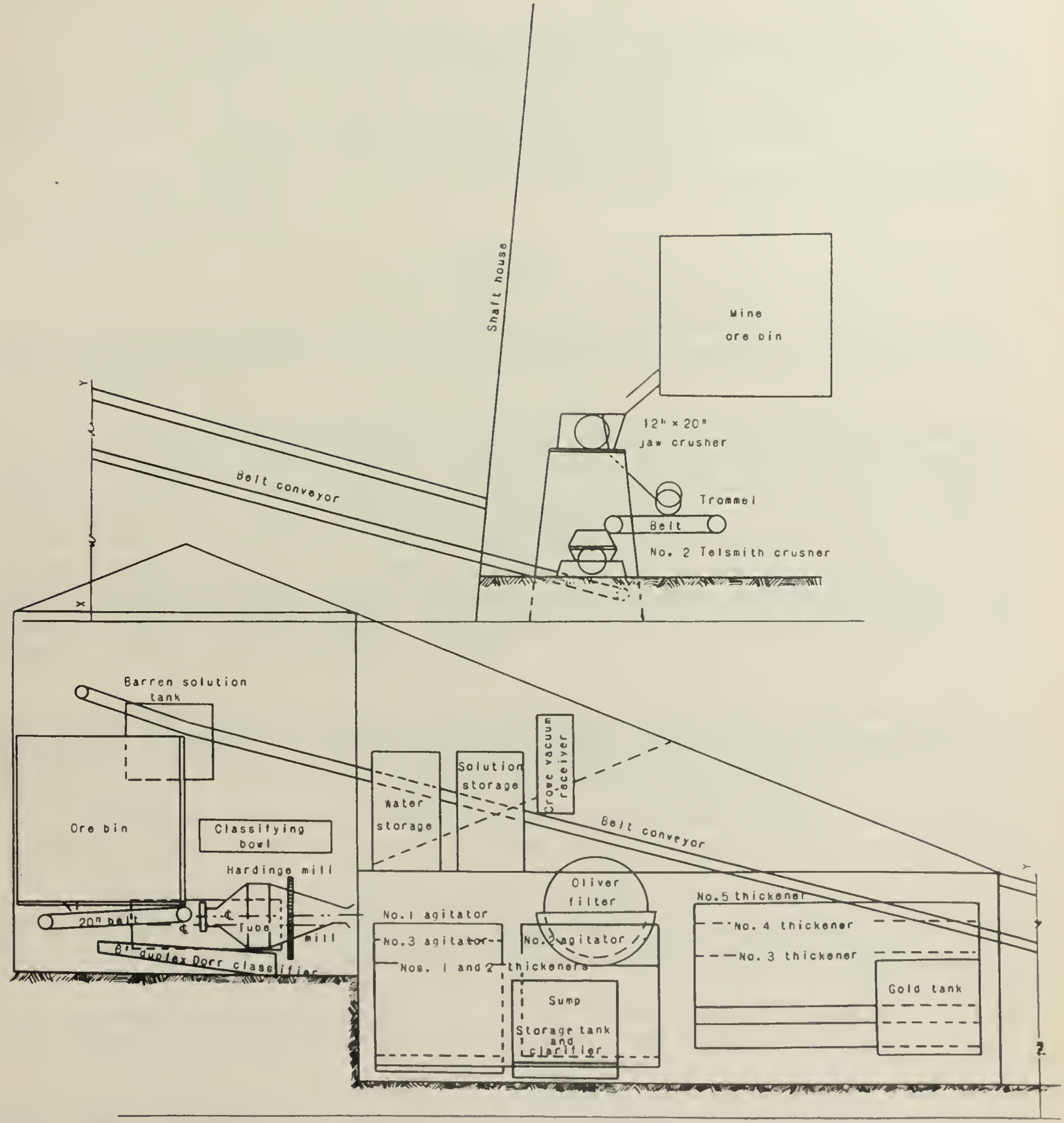

Figure 2.- End elevation of mill

the mill. The travel in the cyanide plant is at right angles to the course through the grinding plant. This results in a compact mill, but does not lend itself to making the present installation a unit in a larger mill.

Overflow from the duplex Dorr classifier goes by gravity to No. 1 thickener. The underflow from this primary thickener is pumped to No. 1 agitator. In No. 1 agitator the pulp is diluted to 70 per cent moisture by the overflow from No. 3 thickener and then pumped by a Wilfley pump to the bowl mounted above the tube mill and No. 1 Classifier.

Average tonnage for the whole mill for 24 hours is 142 tons. The greatest tonnage for any 24 hours is 200 tons.

Breaking

The ore is raised from the mine in 1-ton cars and dumped on a bulldozing platform and bulldozed through grizzly bars, spaced 6 inches apart by spacers 3 feet apart, into a 300-ton coarse-ore bin.

The ore from the coarse-ore bin is fed by gravity to a 12 by 20 inch Buchanan jaw crusher (see fig. 2). This crusher is set with about a 3-inch discharge opening and discharges on a revolving trommel 3 feet in diameter with 1½-inch round openings. The undersize goes by gravity to the main conveyor, and the oversize is delivered by a short belt conveyor to a No. 2 Telsmith crusher set to deliver a product equivalent in fineness to the undersize from the trommel, which it joins on the main conveyor. The head pulley of the oversize conveyor is a magnetic pulley to remove tramp steel from the feed to the Telsmith crusher. The discharge opening of the jaw crusher is so adjusted that the oversize from the trommel just keeps the Telsmith crusher running to capacity.

Operating capacity of the crushing plant as a unit is about 20 tons per hour. However, two 8-hour shifts are often used on the crushers because of lack of capacity in the coarse-ore bin available without resorting to hand shoveling for unloading the bin.

By means of an electric signaling system, the crusherman, ball-mill man, and solution man can be readily called to assist each other in any part of the mill or crusher house.

Manganese-steel jaw plates weigh 420 pounds each and wear about six months. Cheek plates wear about two years. Cast-iron toggles wear about one year. Cast-iron toggles were used after it was found that a great deal of breakage resulted with cast-steel toggles.

Manganese-steel mantles for the Telsmith crusher weigh 560 pounds and wear one year. Manganese-steel concaves weigh 470 pounds per set and also wear one year.

The machines are belt driven from a countershaft which is also belt driven from a 40-hp. motor. Overloads are taken care of by belt slippage, and fuses and overload relays on the motor.

Primary Grinding

Primary grinding is done in one 8 foot diameter by 30-inch Hardinge ball mill running at 16 r.p.m. The rated speed of this mill is 21 r.p.m., but as the full capacity of

the mill is not required, the mill is run at the reduced speed with reduced capacity. This mill runs in an open circuit with no circulating load. However, there is a 12-inch diameter by 3-foot trommel with 3/8-inch round openings bolted to the discharge bell of the mill. About a ton per day of tramp oversize is removed from the mill discharge and returned to the mill by hand. The ball mill discharge averages 75 per cent solids.

The average hourly capacity in 1930 was 6.73 tons, and the ball mill operated 89 per cent of the time. Both the running time and capacity of the mills are limited by the settling capacity of the decantation plant. Table 1 shows screen analyses of the mill products.

Five-inch forged-steel balls are used, manufactured by the Hull Iron and Steel Co. of Hull, Canada. Ball charge is about 30,000 pounds.

Hardinge plate and wedge liners of manganese steel are used. Ball consumption averages 1.96 pounds per ton of ore. Liner consumption averages .24 pound per ton of ore.

The mill is belt driven at 30-foot centers through a spur gear with a gear ratio of 5 to 1, from a 125-hp. squirrel-cage motor. Starting overload is taken care of by a Smith-type Hill friction clutch.

Secondary Grinding

Secondary grinding is done in one 5½-foot diameter by 16-foot power and mining tube mill running at 28 r.p.m. This mill runs in closed circuit with 6 by 20-foot Dorr classifier, and also with a classifying bowl, as shown in the flow sheet and mentioned in the historical review.

The total feed to the tube mill is 12.9 tons per hour. Of this, 6 tons per hour is underflow from the bowl and 6.9 tons per hour return sands from classifier. Assuming that 90 per cent of the total ball-mill feed goes through the tube mill, the circulating load is 2.15 times its original feed.

The tube-mill discharge pulp is kept at about 73 per cent solids; the classifier overflow at about 25 per cent, and the bowl overflow at about 18 per cent solids.

One and one-half inch balls are used, forged from the tops of old railroad rails, by the Burlington Steel Co. of Hamilton, Ontario. The mill is designed for a ball load of 30,000 pounds. The actual ball load has not been determined, but it is probably about half loaded. Ball consumption averages 2.5 pounds per ton. Liner consumption averages .26 pound per ton.

A modified El Oro type of chilled white iron liner is used, manufactured by the Cobalt Foundry of Cobalt, Ontario. Figure 3 shows a drawing of this liner.

The mill is belt driven by a 100-hp. squirrel-cage motor at 30-foot centers through a spur gear with a gear ratio of 5 to 1. Starting overloads are taken care of by a Smith-type, Nill friction clutch.

The Dorr classifier has a grade of 2-1/8 inches per foot and makes 12 strokes per minute.

Cyanidation

The flow sheet of the cyanide plant is shown in Figure 1. The method in use is all sliming in cyanide solution and continuous countercurrent decantation followed by filtration. This system is a development due to changing conditions and the progress of the art. This has been discussed quite fully in the historical review.

An immense amount of research has been done in the effort to devise a more effective treatment, both by ourselves and by other mines in the district. Flotation (either of raw ores or tailings) has not yet been applied with commercial success. Extremely fine grinding is necessary to free values even for flotation, and after the concentrates are made, the commercial recovery of the gold from the concentrates presents a difficult problem. We have tried adding fresh cyanide to the ball mill, to the tube mill, to the primary agitator, and to the secondary agitator. It makes no measurable difference where the cyanide is added as long as the strength is maintained. We now add fresh cyanide to No. 1 agitator.

Grinding solution is kept at 0.9 pound. NaCN equivalent and the other solutions stay at about 0.7 pound per ton. Increasing the strength of cyanide increases both mechanical and chemical loss without affecting the extraction. Decreasing cyanide strength does not necessarily decrease extraction, but when the attempt has been made to carry a lower cyanide strength, it has occasionally resulted in high tailings assays. While it is not certain that the assays were affected by low cyanide, the saving of cyanide by using a strength below 0.9 pound per ton of solution was so small as not to be worth while.

Lime is added at the ball mill and all solutions are saturated with $Ca(OH)_2$. The high lime is necessary for maximum rate of settling and presumably helps to break down tellurides.

No. 1 thickener underflow averages about 58 per cent solids. This is much heavier than the other thickener underflows because, as seen from the flow sheet, the No. 1 thickener has a large, comparatively coarse circulating load. No. 1 agitator is carried at 25 per cent solids and like No. 1 thickener carries a heavy circulating load. This results in giving a preferential agitation to the heavier and higher-grade particles of the ore.

The other agitators are kept at about 18 per cent solids, and the other thickener underflows at about 45 per cent solids.

The total time of treatment is indefinite, due to the preferential treatment and variation in the settling rate, but a change in grade of ore is usually reflected in the tailings after about 36 hours.

The displacement of solution in the thickeners, which are of continuous, rim-launder type, is about 2 tons of solution per ton of ore. The solution displacement on the filter is about 1/2 ton of solution per ton of solids.

Air is introduced incidentally in the air lifts of the Dorr agitators but they are not at all effective as aerators. The solution pumped from the storage pump to the upper storage tank is raised 12 feet above the tank and allowed to plunge into it. Oxygen tests show that this is a very effective way of introducing oxygen. Kieselguhr diffusers are placed in the barren solution tank, in the storage tank, and in the overflows from the thick-

eners. These are fed with compressed air at about 5 pounds pressure, and are very effective aerators. Sodium peroxide and barium peroxide have been tried in various ways and at various times, but with no effect.

Butters-type filter leaves are used to clarify the pregnant solution for precipitation.

The tailings are filtered by one 12 by 12 foot Oliver drum filter. In the filtering cycle 1/4 of the time is used in loading, 3/10 in barren wash, 1/5 in water wash, 1/12 in blow-off and 1/6 is blank. About 0.3 ton barren wash and about 0.2 ton water wash are used per ton of solids. Blow-off air is taken from the mill compressor through a reducing valve at 3 pounds pressure. Vacuum is maintained at 23 to 24 inches of mercury. Twenty-ounce cotton twill is the filtering medium, and the capacity averages 700 pounds of solids per day per square foot of filtering medium with the thin pulp fed to the filter. Cake is discharged containing 80 per cent of solids.

Analysis of pregnant solution is shown in Table 4.

Pumping

Pumping is not a serious problem and has been mentioned incidentally in previous discussions. Slime pumping is done with Barnes 4-inch diaphragm pumps for which we have designed our own valves. The precipitation pump is an Aldrich 5 by 6 inch triplex pump of 100 gallons per minute capacity. Other solution pumps are small centrifugals. Vacuum for the clarifier leaves is furnished by an 8 by 10 inch Gould geared wet vacuum pump. Vacuum for the Oliver filter is supplied by an Oliver type M. S. 14 by 8 inch vacuum pump. The centrifugal pump taking filtrate from the filter has a gland especially designed by the Oliver company to maintain suction against a high vacuum. A 6 by 4 inch single-cylinder, high-speed, Ingersol Rand vacuum pump maintains 26 inches of vacuum for the Crowe vacuum system.

Precipitation

Gold is precipitated from the clarified, deaerated pregnant solution by means of zinc-dust. The solution averages about $4 per ton in gold. Zinc-dust is fed by means of a slow moving 3/4-inch auger in the bottom of a hopper. The zinc drops into a cone which is kept full of barren solution, the supply of which is regulated by means of a float valve. The emulsion of barren solution and zinc is pumped from the bottom of the cone by a 12½ by 4 inch Gould single-cylinder geared pump to a small receiver near the presses where it joins the pregnant solution from the precipitation pump.

Originally the zinc was fed into the suction of the precipitation pump, but after the installation of the Crowe vacuum system, precipitates lodged in passages of the pump and in the pipe lines and were very difficult and expensive to dislodge; they caused difficulty in checking recovery and occasioned loss of precipitates in cleaning pipe lines.

Two square Perrin plate and frame filter presses, 24 by 24 inches in size are used alternately, dressed with 10-ounce duck covered by filter paper. In normal operations except for three hours at the start of a new press the barren solution contains only a trace of gold. Ninety-nine per cent of the gold in the pregnant solution is precipitated.

During 1930, 1.9 ounces of zinc was used per ounce of gold and silver recovered.

Lead nitrate is fed to No. 1 agitator at the rate of 1½ pounds per day. This prevents fouling of solutions and promotes good precipitation. Although precipitation may remain good for months at a time, if no lead is used, yet when for any reason precipitation does become bad, it is a great deal worse if there is no lead in the solutions.

Refining

Eighteen hours before a press is to be cleaned up, the solution is turned off and compressed air is turned into the press. After blowing for 18 hours the air is turned off and the press opened and cleaned. The canvas is left to use again and the paper separated from the precipitates and burned separately in the furnace. The long blowing with air oxidizes a great deal of the excess zinc left in the press and leaves the precipitates quite dry.

In the refinery the precipitates are mixed with flux by shoveling over and over in a steel mixing box. The following charge is used:

	Pounds
Precipitates..........	100
Manganese dioxide	4
Ferric oxide..........	4
Crude soda.............	6
Air-slake lime......	16
Sodium nitrate......	10
Borax glass...........	25
Silica sand...........	25

The thoroughly mixed charge is shoveled into an oil-fired Rockwell tilting furnace 2 feet 6 inches in diameter by 3 feet 6 inches long, outside dimensions. The furnace has a rammed carborundum lining, sold under the trade name of "carbofrax." We drilled a number of 3/4 inch holes in the steel shell of this furnace to make it easier to break out an old worn-out lining from the outside.

After a charge comes to a quiet fusion fresh charge is added and the furnace is heated again; the operation is repeated until the furnace is full of molten slag. Slag is then poured into a conical slag pot, and a fresh charge is added. This procedure is continued until the precipitates are all melted. Most of the slag is then poured into slag pots, after which the gold is also poured into slag pots, care being taken that the metal buttons are not too large to handle and to charge back into the furnace.

The buttons, when cool, are freed from adhering slag and charged back into the furnace and melted. Sodium nitrate is thrown on the pool of molten metal. This reacts with the base metals, forming fumes and a crust of metallic oxides. The crust is skimmed off and the metal is heated again. This procedure is repeated until bullion of the desired fineness is indicated by the metal hardening at a high temperature. The bullion is then poured into bullion molds for shipment.

Skimmings and slag from the buttons and bricks are added to the next clean-up. The rest of the slag is returned to the ball mill. Investigation has shown that all the gold in the slag is easily dissolved in the mill and that retreating the slag in the mill does not foul the solutions. When a lining is removed, any coarse pieces of gold are picked out by hand and added to the next melt. The scrap lining is thrown into the ball mill. The gold in the lining is easily dissolved and the lining material has no precipitating effect on the solutions. As lining and slag are impossible to sample accurately, the refinery receives no credit for the gold thus returned to the mill. There is a small but undetermined loss of gold in fumes; and there are no other refinery losses except the mechanical loss that is unavoidable when handling such high-grade material. The bullion produced varies from 930 to 955 parts in a thousand total fineness of gold and silver. Precipitates are usually about 50 per cent fine gold and silver.

Launders

Pulp launders throughout the mill have a grade of 1 inch per foot. The discharge launders from the ball mill and from the tube mill are of wood-lined with steel plate. All other launders are wood, unlined.

Tailings Disposal

Tailings are diluted with water to a ratio, water-to-solids, of 4 to 1 and are pumped by a Wilfley 3-inch sand pump through 6-inch, inside diameter, wood stave pipe, 2,800 feet to Kirkland Lake. The pipe line is laid out carefully on a grade of 1/4 inch to the foot and supported on trestles. Extreme care was taken to leave no low places, so that the pipe drains itself and does not freeze up in cold weather when the flow stops for any reason.

Sampling and Estimating Recovery

Head sample.- At 2-hour intervals a cut is taken from the ball feed belt, 1 foot in length and across the belt. This ore is weighed and returned to the mill. The revolutions of the head pulley of the conveyor are counted by an automatic counter. The average of the "belt weights" multiplied by the revolutions of the head pulley and multiplied by a constant equals the tons of ore fed to the mill. The constant includes the circumference of the head pulley, the thickness of the belt, the conversion of pounds to tons, and a correction for the average moisture of the ore.

At 2-hour intervals (all samples are taken by hand and at 2-hour intervals by cutting the full pulp stream with a sampling dipper, which is emptied into a sample pail) a sample is taken from the discharge of the ball mill. At the same time a specific-gravity determination is made of the ball-mill discharge and a sample is taken of the storage solution entering the ball mill. The ball-mill discharge sample at the end of the mill day is thoroughly mixed and quartered down wet. The quartered down sample weighing about 250 grams is filtered and washed with water, making about 600 c.c. of filtrate. The filtered pulp is weighed, dried, and assayed. The filtrate is measured and assayed.

The dissolved gold in the ball-mill discharge equals

$$\frac{\text{Number of cubic centimeters of filtrate X assay of filtrate.}}{\text{Number of grams of solids}}$$

The gold entering the ball mill in the grinding solution equals

$$\frac{\text{Per cent of moisture in ball-mill discharge}}{\text{Per cent of solids in ball-mill discharge}}$$

multiplied by the assay of the storage solution. The gold value of the plant heads is the undissolved gold plus the dissolved gold in the ball-mill discharge minus the gold entering the ball mill in grinding solution per ton of ore. The gold value multiplied by the number of tons gives the amount of gold entering the plant for the day.

This rather involved procedure has been found necessary for two reasons:- First, it is impossible to get a consistent, reliable sample of the ore entering the mill without expensive sampling equipment to reduce a very large sample. Second, evaporating a pulp containing gold in solution causes segregation of the gold in the pulp and on the dish, making the sample entirely unreliable.

Tailings Sample.- The tailings sample is taken by hand, quartered down wet to about 600 grams of solids, and filtered and washed with water to make about 600 c.c. of filtrate. Then the

$$\frac{\text{Number of cubic centimeters of filtrate x assay of filtrate}}{\text{grams of solids}}$$

equals dissolved gold loss per ton, which added to the assay of the solids, gives the total loss of gold per ton. It is assumed that during each day the quantity of solids discharged equals that entering the mill as ore. This is a false assumption, but over a month's time it approximates the truth; and the number of tons milled multiplied by the grade of the tailings approximates the loss of gold in the tailings. The difference between the gold entering the plant and the gold discharged is estimated as the recovery. Over a period of years this estimate agrees almost exactly with bullion recovery. On a month's run it may vary as much as 15 per cent, partly due to varying amounts of gold trapped in classifiers, launders, and in the tube-mill lining.

Press Recovery Estimate.- A drip sample is taken from the pipe entering the gold solution tank and is called "press heads." A drip sample is taken from the discharge solution from the press and is called "press tails." Press heads minus press tails multiplied by number of tons precipitated is the estimated press recovery. Once during each shift the flow of clarified solution to the gold tank is stopped for 30 minutes. The number of inches of solution precipitated out of the tank is measured, and it is assumed that the precipitation takes place at the same rate throughout the shift. From the diameter of the tank and the number of inches pumped per hour, the solution tonnage precipitated is calculated. The press recovery is calculated from number of tons precipitated multiplied by the difference between press heads and press tails. Over a period of years this checks very closely with bullion recovery. For a year it checks within 1 per cent. For one month there may be a variation of 10 per cent, partly due to gold absorption in the furnace lining.

Other Samples.- Besides the above samples a sample is regularly taken to show the dissolved gold going to the filter and a sample of the underflow and overflow of the bowl. The bowl samples are quartered down, filtered, and washed to determine the undissolved gold.

WATER SUPPLY

Mine water is used for water wash on the filter. It is first used as cooling water in the mine compressors and thus materially reduces the cost of heating the mill. One-fifth of a ton of water is used on the filter per ton of ore and just replaces the solution carried out by the discharged cake. The mine water, while rather hard, carries no harmful impurities. Water for tailings disposal amounting to about 1,000 gallons per ton of ore, is partly mine water and partly purchased from the municipal plant of Kirkland Lake.

LABOR

Seven men operate the plant, working 8-hour shifts, as follows:

	Cost per shift
Three solution men, at..	$5.00
Three ball mill men, at	4.50
One crusher man, at........	4.00
Average daily wage....	4.64

Table 1.- Screen sizes

Kirkland Lake Gold Mining Co., Ltd.

Screen	Ball-mill feed	Ball-mill discharge	Tube mill discharge	Classifier rake return	Classifier over-flow	Bowl under-flow	Bowl over-flow
Minus 2 inch ring plus 1 inch square	31.6	-	-	-	-	-	-
Minus 1 inch square plus ½ inch square	30.7	-	-	-	-	-	-
Minus ½ inch square plus 3 mesh	13.8	-	-	-	-	-	-
Minus 3 mesh plus 10 mesh	13.7	2.0	-	2.5	-	-	-
Minus 10 mesh plus 20 mesh	4.1	7.3	-	10.0	-	-	-
Minus 20 mesh plus 40 mesh	2.5	17.5	0.5	12.5	-	-	-
Minus 40 mesh plus 65 mesh	1.4	20.4	1.5	13.0	-	-	-
Minus 65 mesh plus 80 mesh	.3	8.1	2.0	7.9	-	-	-
Minus 80 mesh plus 100 mesh	.3	7.5	7.2	10.7	-	-	-
Minus 100 mesh plus 150 mesh	1.6[1]	5.1	9.7	14.1	-	-	-
Minus 150 mesh plus 200 mesh	-	4.0	17.0	10.7	32.0	47.5	1.0
Minus 200 mesh plus 300 mesh	-	28.3[1]	62.1[1]	18.6[1]	8.2	22.8	3.0
Minus 300 mesh	-	-	-	-	59.8	29.7	96.0

01 - Smallest screen used.

Note:- The analyses in this table are not averages but are typical of operations in the last six months of 1930.

Table 2.- Average mill assays

Kirkland Lake Gold Mining Co., Ltd. Period covered: Year of 1930

Product	Gold, dollars	Gold, ounces
Average heads	11.38	0.553
Average tailings	1.22	.0593
Pregnant solution	3.86	.1877
Barren solution	.044	.0022
Dissolved loss	.0661	.00322
Bowl overflow undissolved	2.19	.1065
Bowl underflow undissolved	4.54	.222
Percentage recovered, 89.27		

Table 3.- Metallurgical data

Kirkland Lake Gold Mining Co., Ltd. Period covered: Year of 1930

Head assay	$11.38
Total ore treated ... tons	52,768
Days operated	365
Hours operated per day	21.4
Operating time ... per cent	89.3
Average tonnage per 24 hours	142.6
Recovery of gold per ton	$10.16
Consumption of cyanide[1] ... pounds	73,000
Consumption of lime ... do.	315,956
Consumption of zinc ... do.	3,718
Consumption of lead nitrate ... do.	520
Soluble gold in tailings per ton of ore	$0.0661
Net water consumption, gallons per ton	1,000
Ball consumption ... pounds	234,377
Liner consumption ... do.	26,200

1 - Aero brand 50 per cent NaCN

Table 4.- Analysis of pregnant solution

Cyanide, as KCN	0.038 per cent
Lime, CaO	.0972 do.
Thiocyanate, as KCNS	.2323 gram per liter
Ferrocyanide, as $K_4Fe(CN)_6$	Nil
Ferricyanide, as $K_6Fe(CN)_6$	Trace
Soluble sulphates, sulphur	.02025 gram per liter
Fe	.0015 do.
Ca	.0130 do.
Pb	.0043 do.
Cu	.0115 do.
Te	Nil
Zn	.032 gram per liter
Au	5.5 mgm per liter
Ag	1.06 do.

1 - Cyanide compounds are reported as potassium salts although not known to be present in that form.

Note:- Reducing power of solution: 1 c.c. requires 0.18 c.c. $\frac{N}{10}$ $KMnO_4$ under standard conditions.

Table 5.- Analysis of tailing

	Per cent
Silica	49.6
Alumina	11.85
Ferric oxide	1.52
Ferrous oxide	6.18
Manganese oxide	.13
Lime	6.8
Soda	6.0
Potash	4.44
Carbon dioxide	9.01
Titanium dioxide	.87
Phosphorus pentoxide	.13
Pyrite	1.84
Molybdenite	None
Lead	Trace
Tellurium	.005
Copper	None
Gold	.0001, or $0.80 per ton
Silver	Trace

Table 6.- Summary of costs

Kirkland Lake Gold Mining Co., Ltd. Period: 1930. Milling method: All sliming.

Ton of ore treated: 52768

Production: Gold, 25774 ounces
Silver, 3593 ounces

	Operating labor	Supervision	Power	Supplies	Reagents	Repair labor	Miscellaneous	Total
Crushing........	$0.0504	$0.01	$0.0263	$0.0401		$0.0139	$0.0043	$0.1450
Grinding........	.0866	.02	.2555	.2342		.0394	.0060	.6417
Cyanidation..	.1093	.03	.0847	.1031	$0.153	.0458	.0063	.5322
Refining......	.004	.01	.0012	.0347		.0004	.0033	.0536
Sampling and assaying....	.0091			.0064			.001	.0165
Total............	.2594	.07	.3677	.4185	.153	.0995	.0209	1.389

Note:- Water is charged as supplies.

No interest, head-office charges, taxes of amortization are included.

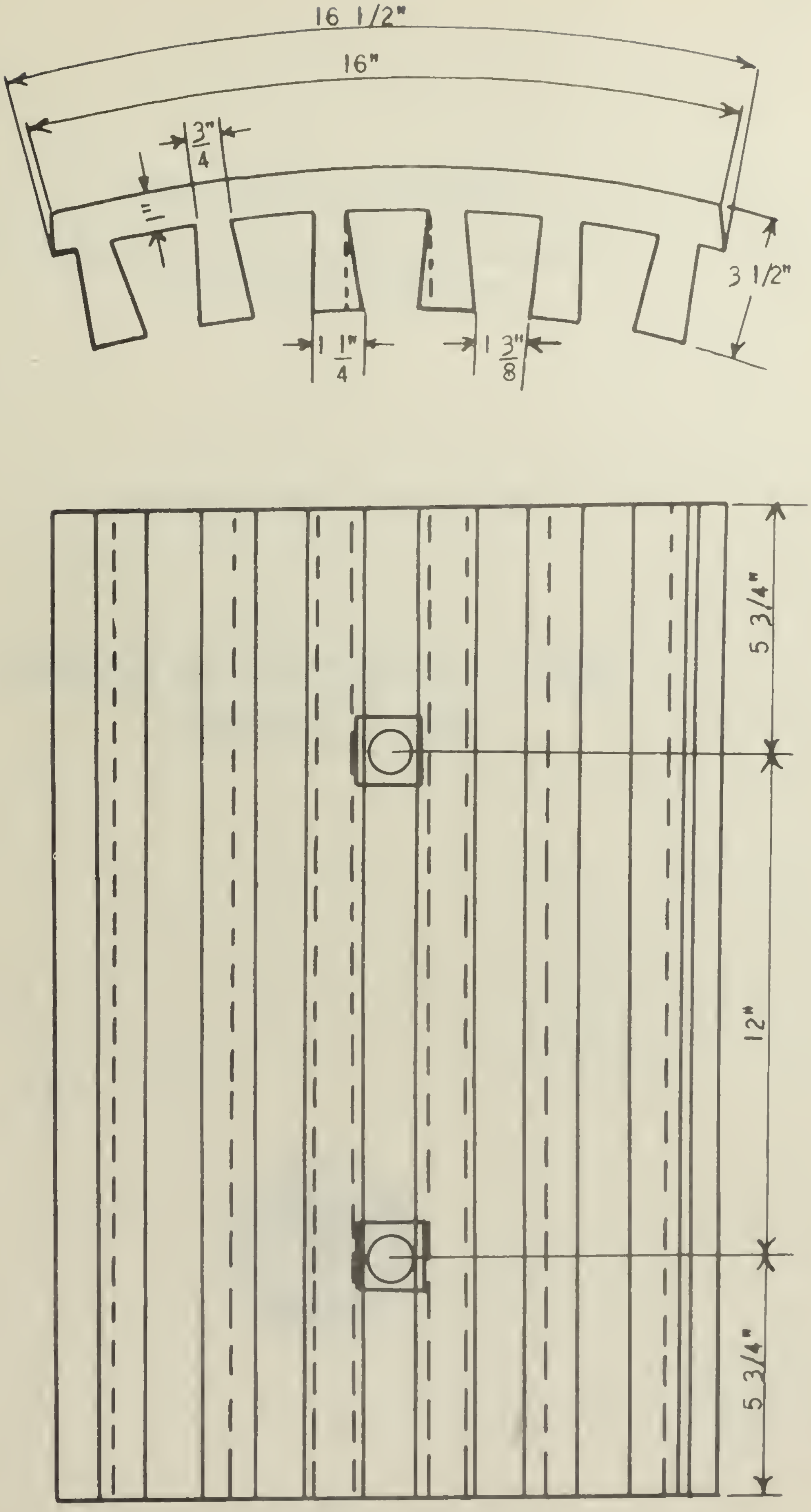

Figure 3.- Shell liner for 5 1/2 by 16 foot tube mill

I. C. 6509

June, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SURVEY OF CRACKING PLANTS

JANUARY 1, 1931

BY

G. R. Hopkins

I. C. 6509
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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SURVEY OF CRACKING PLANTS, JANUARY 1, 1931[1]

By G. R. Hopkins[2]

Introduction

According to reports as of January 1, 1931, the total daily charging capacity of the cracking plants, both completed and under construction, in the United States amounted to 1,950,781 barrels, an increase over the previous year of 14 per cent. The total daily capacity of the operating plants amounted to 1,594,990 barrels, or 82 per cent of the grand total, the capacity of the inoperative plants was 244,661 barrels, or 12 per cent of the total, while the capacity of the plants under construction amounted to 111,130 barrels, or 6 per cent. In comparison with a year ago, these data indicate a decline in the total capacity of the plants being built but show material increases in the capacity of both the other two classes. The increase in total inoperative capacity and the decline in construction in 1930 undoubtedly resulted from unsatisfactory economic conditions.

The total number of cracking units completed or under construction on January 1, 1931, totaled 1,868 as compared with 2,002 the previous year and with a high of 2,559 on June 1, 1926. During the period between June 1, 1926, and January 1, 1931, the total charging capacity of the cracking plants has more than doubled, but the number of units has shown a material decline. This condition has been brought about largely through the dismantling of many Burton stills of small capacity and through the tendency to increase the size of the new units. The largest unit that can be identified in this survey has a daily charging capacity of 10,000 barrels, but undoubtedly there are some that would exceed this in size.

Texas, with a total of 575,300 barrels, ranked far ahead of the other States in charging capacity, and was followed in order by California, Indiana, Oklahoma, and Pennsylvania. Indiana, with only five refineries, ranked third in cracking capacity, which illustrates the importance of the Chicago district as a gasoline-producing center. Not all of the States showed a gain in capacity during the year; in fact, four of them, namely Illinois, Louisiana, New York, and Oklahoma, reported a decline.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6509."
2 Economic analyst, U. S. Bureau of Mines.

The production of gasoline by means of the cracking process has shown a steady increase since such statistics were first compiled (1925). In 1930 the production by this method amounted to 164,243,000 barrels, an increase over 1929 of 20,516,000 barrels, or 14 per cent. As the increase in total motor production in 1930 was only 0.5 per cent, the relative proportion of cracked gasoline rose from 33.0 per cent in 1929 to 37.7 per cent in 1930. In December, 1930, the ratio of cracked gasoline to the total reached 41.1 per cent, the first month that it had exceeded 40 per cent.

The Texas district led in the production of cracked gasoline in 1930, although the East Coast district retained first place from the standpoint of percentage, that is, in the proportion of cracked gasoline to the total. Nearly half of the gasoline produced by the refineries on the Atlantic seaboard is made by the cracking process. The production of cracked gasoline at California refineries showed a material increase in 1930, although the proportion to the total amounted to only 18 per cent, the lowest of any of the major refining districts. The Louisiana Gulf Coast district made the most notable gain in cracking activity; its percentage rose from 30 in 1929 to 45 in 1930.

Thirty-five different types of cracking processes are listed in this survey, as compared with 38 a year ago. Of the 35 types on January 1, 1931, only 6 were being actively licensed, 6 were in use at only a few refineries, several were practically obsolete but still in use, and a considerable number were being developed by one company. Where a cracking process is being developed by one company but no particular name has been assigned to it, the term "Own" is used. The number of processes being developed by one company has shown a rapid increase in the last few years. The development of pipe stills which can be used for either skimming or cracking indicates that the dividing line between these two refining methods will tend to disappear.

The year witnessed a further gain in importance of the vapor-phase method of cracking. The total daily capacity of the vapor-phase units completed or under construction rose from 30,650 barrels on January 1, 1930, to 45,600 barrels on January 1, 1931. The number of companies using this method rose from 10 to 13 in the same period.

The growth in the utilization of refinery or still gases was continued throughout 1930. A number of refineries installed vapor recovery systems to collect these gases, which had been going to waste. The major part of the gases recovered are treated in absorption plants and the gasoline obtained has constituted an appreciable item in the motor-fuel supply. The residue gas is turned into pipe lines or is used for fuel under boilers and stills. A material factor in the increased supply of refinery gas has been a gain in the use of vapor-phase cracking processes which produce large quantities of such gas. The increase in vapor-phase cracking has been due primarily to an increased demand for antiknock motor fuel, but it is probable that some installations have been influenced by the fact that there was a ready market for the gas. Complete information as to the production of gas by cracking plants is not available, but it is probable that the total output during 1930 amounted to approximately 200 billion cubic feet, an increase over 1929 of 15 per cent.

RECAPITULATION BY TYPE OF PROCESS

Type of process	January 1, 1931		January 1, 1930	
	Total units	Total capacity, barrels per day	Total units	Total capacity, barrels per day
Black	12	16,000	12	16,000
Burton	793	164,249	895	171,440
Carborundum	1	3,000	1	3,000
Coil cracker	2	38,000	-	-
Continuous high-pressure	2	5,300	2	5,300
Controlled coil	2	4,500	-	-
Convertor	1	4,200	1	1,500
Cross	150	245,800	163	273,670
de Florez	6	13,550	1	1,200
Doherty	20	27,500	16	20,200
Donnelly	4	7,000	2	3,000
Dubbs	185	252,250	188	238,500
Fleming	2	800	8	2,700
Gyro	20	16,000	13	9,200
Holmes-Manley	115	233,900	129	240,800
Isom	115	179,150	100	100,000
Jenkins	46	66,150	49	75,000
Leamon	5	2,250	5	2,250
Lewis	4	6,200	4	6,200
Lientz	1	6,000	1	6,000
Link	22	46,000	20	50,000
Louisiana coil	5	7,100	-	-
Ormont	4	1,000	4	1,000
Own	166	168,000	186	151,000
Pipe stills	3	13,000	1	3,500
Pratt	2	2,500	1	500
Pressure coke	13	22,000	13	19,500
Richmond	2	6,400	2	6,000
Skelly-Rittman vapor-phase	2	2,500	-	-
Slagter	18	2,700	24	3,600
Snodgrass	14	4,972	24	5,680
True vapor-phase	2	3,600	2	10,500
Trumble	1	750	1	750
Tube-and-tank	118	385,460	123	260,100
Vapor-phase	3	4,200	3	6,000
Winkler-Koch	7	18,800	5	10,500
Other	-	-	3	4,350
Total	1,868	1,950,781	2,002	1,708,940

RECAPITULATION BY YEARS

Year	Total units	Charging capacity, barrels per day			
		Operating	Shut down	Building	Total
June 1, 1925	2,527	690,492	26,200	116,000	832,692
June 1, 1926	2,559	844,800	47,690	47,600	940,090
Jan. 1, 1928	2,334	1,013,000	253,000	22,000	1,288,000
Jan. 1, 1929	2,205	1,194,501	147,923	134,450	1,476,874
Jan. 1, 1930	2,002	1,419,200	139,840	149,900	1,708,940
Jan. 1, 1931	1,868	1,594,990	244,661	111,130	1,950,781

RECAPITULATION BY DISTRICTS

District	Total units	Charging capacity, barrels per day			
		Operating	Shut down	Building	Total
East Coast	158	234,200	28,000	43,100	305,300
Appalachian ...	64	45,522	6,500	6,250	58,272
Ind., Ill., Ky., etc.	435	245,930	56,900	32,880	335,710
Okla., Kans., etc.	355	228,550	29,250	8,000	265,800
Texas	273	525,100	47,600	2,600	575,300
La. and Ark. ..	79	90,400	15,950	6,000	112,350
Rocky Mountain.	427	44,538	57,461	2,300	104,299
California	77	180,750	3,000	10,000	193,750
U. S., total.	1,868	1,594,990	244,661	111,130	1,950,781
Texas Gulf Coast	203	434,700	33,600	2,600	470,900
La. Gulf Coast.	39	64,000	3,000	6,000	73,000

RECAPITULATION BY STATES

State	Total units	Charging capacity, barrels per day			
		Operating	Shut down	Building	Total
Arkansas	25	8,900	8,550	-	17,450
California	77	180,750	3,000	10,000	193,750
Colorado	12	2,322	548	800	3,670
Georgia	2	3,600	-	-	3,600
Illinois	114	78,200	11,000	2,500	91,700
Indiana	252	126,850	45,900	13,000	185,750
Iowa	1	-	1,000	-	1,000
Kansas	87	93,700	10,850	-	104,550
Kentucky	37	15,480	-	2,380	17,860
Louisiana	54	81,500	7,400	6,000	94,900
Maryland	22	21,400	3,000	8,600	33,000
Massachusetts..	16	25,300	6,000	-	31,300
Michigan	3	3,600	-	-	3,600
Missouri	86	15,000	8,400	4,000	27,400
Montana	4	3,300	500	1,500	5,300
New Jersey	44	78,050	10,250	17,200	105,500
New York	11	11,600	-	-	11,600
Ohio	43	39,650	800	15,750	56,200
Oklahoma	181	119,850	9,000	4,000	132,850
Pennsylvania ..	93	104,172	4,900	22,050	131,122
Rhode Island ..	3	6,000	-	-	6,000
South Carolina.	8	5,250	8,750	-	14,000
Texas	273	525,100	47,600	2,600	575,300
Utah	32	6,400	3,800	-	10,200
West Virginia..	9	6,500	800	750	8,050
Wyoming	379	32,516	52,613	-	85,129
U. S., total.	1,868	1,594,990	244,661	111,130	1,950,781

SURVEY OF CRACKING PLANTS, JANUARY 1, 1931

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	ARKANSAS				
SD	Houston Oil Co. of Texas	Camden	2	2,000	Dubbs
SD	Kettle Creek Refg. Co.	El Dorado	2	2,000	Dubbs
Op.	Lion Oil Refg. Co.	El Dorado	10	4,500	Burton
Op.	Lion Oil Refg. Co.	El Dorado	1	2,000	Own
SD	Lion Oil Refg. Co.	El Dorado	5	2,250	Burton
SD	Ouachita Valley Refg.Co.	El Dorado	1	800	Dubbs
SD	Root Refg. Co.	El Dorado	2	1,500	Dubbs
Op.	Simms Oil Co.	Smackover	2	2,400	Cross
			25	17,450	
	CALIFORNIA[1]				
Bldg.	Associated Oil Co.	Avon	1	10,000	Tube-and-tank
SD	General Pet. Corp. of Calif.	Los Angeles	2	3,000	Vapor-phase
Op.	Hercules Gasoline Co.	Los Angeles	1	1,250	Jenkins
Op.	Richfield Oil Co. of Calif.	Hynes	4	14,000	Cross
Op.	Richfield Oil Co. of Calif.	Watson	12	16,000	Black
Op.	Rio Grande Oil Co.	Vinvale	2	2,000	Jenkins
Op.	Shell Oil Co.	Dominguez	8	32,000	Dubbs
Op.	Shell Oil Co.	Martinez	8	12,000	Dubbs
Op.	Shell Oil Co.	Martinez	1	4,200	Convertor
Op.	Shell Oil Co.	Watson	8	15,600	Dubbs
Op.	Standard Oil Co. of Calif.	El Segundo	12	36,300	Dubbs
Op.	Standard Oil Co. of Calif.	Richmond	8	18,200	Dubbs
Op.	The Texas Co.	Fillmore	1	2,700	Cross
Op.	The Texas Co.	Watson	3	5,500	Holmes-Manley
Op.	Union Oil Co. of Calif.	Wilmington	4	12,000	Cross
Op.	Union Oil Co. of Calif.	Wilmington	1	6,000	Lientz
Op.	Union Oil Co. of Calif.	Wilmington	1	3,000	Carborundum
			77	193,750	

1/ Data compiled by E. T. Knudsen of the San Francisco Office of the U. S. Bureau of Mines.

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	COLORADO				
Bldg.	Continental Oil Co.	Denver	1	800	Cross
Op.	Continental Oil Co.	Florence	6	822	Burton
SD	Continental Oil Co.	Florence	4	548	Burton
Op.	The Texas Co.	Craig	1	1,500	Holmes-Manley
			12	3,670	
	GEORGIA				
Op.	The Atlantic Refg. Co.	Brunswick	2	3,600	Lewis
			2	3,600	
	ILLINOIS				
Op.	The Globe Oil & Refg. Co.	Lemont	1	3,000	Winkler-Koch
Op.	Indian Refg. Co.	Lawrenceville	8	8,000	Cross
Op.	Lincoln Oil Refg. Co.	Robinson	4	7,600	Holmes-Manley
Bldg.	Lubrite Refg. Co.	E. St. Louis	2	2,500	Pratt
Op.	Shell Pet. Corp.	Wood River	14	14,000	Dubbs
SD	Shell Pet. Corp.	Wood River	2	2,000	Dubbs
Op.	Shell Pet. Corp.	Wood River	4	3,000	Cross
Op.	Shell Pet. Corp.	Wood River	1	3,000	True vapor-phase
Op.	Standard Oil Co. (Ind.)	Wood River	20	6,500	Burton
SD	Standard Oil Co. (Ind.)	Wood River	40	9,000	Burton
Op.	Standard Oil Co. (Ind.)	Wood River	6	13,000	Holmes-Manley
Op.	The Texas Co.	Lockport	6	9,000	Holmes-Manley
Op.	The Texas Co.	Lockport	1	3,000	de Florez
Op.	The Texas Co.	Lockport	2	5,000	Pressure-coke
Op.	White Star Refg. Co.	Wood River	3	3,100	Dubbs
			114	91,700	
	INDIANA				
Op.	Bartles-Maguire Oil Co.	East Chicago	2	2,000	Jenkins
Op.	Bartles-Maguire Oil Co.	East Chicago	1	1,200	Vapor-phase
Op.	Empire Oil & Refg. Co.	East Chicago	10	15,000	Doherty
Op.	Shell Pet. Corp.	East Chicago	8	12,000	Dubbs
Op.	Sinclair Refg. Co.	East Chicago	33	45,350	Isom
SD	Sinclair Refg. Co.	East Chicago	5	6,500	Isom
Bldg.	Sinclair Refg. Co.	East Chicago	3	13,000	Isom
SD	Standard Oil Co. (Ind.)	Whiting	166	29,900	Burton
Op.	Standard Oil Co. (Ind.)	Whiting	17	46,000	Holmes-Manley
SD	Standard Oil Co. (Ind.)	Whiting	5	9,500	Holmes-Manley
Op.	Standard Oil Co. (Ind.)	Whiting	2	5,300	Continuous high-pressure
			252	185,750	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	IOWA				
SD	MonaMotor Oil Co.	E. Omaha	1	1,000	Cross
			1	1,000	
	KANSAS				
Op.	Barnsdall Refineries, Inc.	Wichita	4	1,500	Dubbs
Op.	Derby Oil Co.	Wichita	3	2,500	Dubbs
Op.	The El Dorado Refg. Co.	El Dorado	1	2,000	Winkler-Koch
Op.	Golden Rule Refg. Co.	Wichita	1	1,200	Jenkins
SD	Hutchinson Oil Refg. Co.	Hutchinson	1	750	Trumble
Op.	Independent O. & G. Co. (Phillips)	Kansas City	1	3,000	Dubbs
SD	Kanotex Refg. Co.	Arkansas City	3	3,600	Jenkins
Op.	Kanotex Refg. Co.	Arkansas City	1	2,500	Donnelly
Op.	National Refg. Co.	Coffeyville	1	1,500	Own
SD	The Peerless Oil & Refg. Co. (Altitude)	Chanute	2	2,000	Jenkins
Op.	Shell Pet. Corp.	Arkansas City	8	7,500	Dubbs
SD	Shell Pet. Corp.	Arkansas City	2	2,000	Dubbs
Op.	Sinclair Refg. Co.	Coffeyville	10	13,000	Isom
Op.	Sinclair Refg. Co.	Kansas City	10	13,000	Isom
Op.	Skelly Oil Co.	El Dorado	9	9,000	Jenkins
Op.	Skelly Oil Co.	El Dorado	3	13,000	Pipe stills
SD	Skelly Oil Co.	El Dorado	2	2,500	Skelly-Rittman, vapor-phase
Op.	The Standard Oil Co. (Kans.)	Neodesha	20	12,000	Burton
Op.	The Standard Oil Co. (Kans.)	Neodesha	1	2,500	Holmes-Manley
Op.	Vickers Pet. Co. of Del.	Potwin	2	1,500	Dubbs
Op.	White Eagle Oil Corp.	Augusta	2	8,000	Coil-cracker
			87	104,550	
	KENTUCKY				
Op.	Aetna Oil Service, Inc.	Louisville	1	600	Fleming
Op.	Ashland Refg. Co.	Leach	1	1,000	Dubbs
Op.	Latonia Refg. Co.	Latonia	1	2,380	Tube-and-tank
Bldg.	Latonia Refg. Co.	Latonia	1	2,380	Tube-and-tank
Op.	Louisville Refg. Co.Inc.	Louisville	2	2,000	Dubbs
Op.	Standard Oil Co. (Ky.)	Louisville	30	7,500	Burton
Op.	The Texas Co.	Pryse	1	2,000	Holmes-Manley
			37	17,860	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	LOUISIANA				
Op.	Chalmette Pet. Corp.	Chalmette	1	2,000	Winkler-Koch
SD	Crystal Oil Refg. Corp.	Cedar Grove	2	3,400	Jenkins
Op.	Louisiana Oil Refg. Corp.	Bossier City	4	6,400	Tube-and-tank
Op.	Louisiana Oil Refg. Corp.	Bossier City	5	7,100	Louisiana coil
Op.	Shell Pet. Corp.	Norco	4	7,000	Dubbs
SD	Shell Pet. Corp.	Norco	2	3,000	Dubbs
Op.	Shreveport-Eldorado Pipe Line Co., Inc.	Shreveport	2	2,000	Dubbs
SD	Shreveport-Eldorado Pipe Line Co., Inc.	Shreveport	1	1,000	Jenkins
Op.	Standard Oil Co. of La.	Baton Rouge	20	40,000	Link
Bldg.	Standard Oil Co. of La.	Baton Rouge	2	6,000	Link
Op.	Standard Oil Co. of La.	Baton Rouge	8	10,000	Tube-and-tank
Op.	Standard Oil Co. of La.	Baton Rouge	2	5,000	Cross
Op.	Stanolind Oil & Gas Co.	Superior	1	2,000	Cross
			54	94,900	
	MARYLAND				
Op.	Continental Oil Co.	Baltimore	2	4,000	Cross
Op.	Continental Oil Co.	Baltimore	4	2,400	Dubbs
SD	Interocean Oil Co.	Baltimore	2	1,000	Dubbs
SD	Interocean Oil Co.	Baltimore	4	2,000	Leamon
Op.	Standard Oil Co. of N.J.	Baltimore	8	15,000	Tube-and-tank
Bldg.	Standard Oil Co. of N.J.	Baltimore	2	8,600	Tube-and-tank
			22	33,000	
	MASSACHUSETTS				
Op.	Cities Service Refg. Co.	E. Braintree	2	4,500	Doherty
SD	Cities Service Refg. Co.	E. Braintree	2	1,800	Holmes-Manley
Op.	Colonial Beacon Oil Co., Inc.	Everett	10	20,800	Tube-and-tank
SD	Colonial Beacon Oil Co., Inc.	Everett	2	4,200	Tube-and-tank
			16	31,300	
	MICHIGAN				
Op.	White Star Refg. Co.	Trenton	3	3,600	Dubbs
			3	3,600	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	MISSOURI				
SD	Joplin Refg. Co.	Joplin	1	1,000	Jenkins
Op.	Standard Oil Co. (Ind.)	Sugar Creek	20	7,000	Burton
SD	Standard Oil Co. (Ind.)	Sugar Creek	60	7,400	Burton
Op.	Standard Oil Co. (Ind.)	Sugar Creek	4	8,000	Holmes-Manley
Bldg.	Standard Oil Co. (Ind.)	Sugar Creek	1	4,000	Holmes-Manley
			86	27,400	
	MONTANA				
SD	Arro Oil & Refg. Co.	Lewistown	1	500	Dubbs
Op.	Hart Refineries	Missoula	1	300	Own
Op.	International Refg. Co.	Sunburst	1	3,000	de Florez
Bldg.	Laurel Oil & Refg. Co.	Laurel	1	1,500	Donnelly
			4	5,300	
	NEW JERSEY				
Op.	The Bertrin Pet. Co.	Maurer	1	3,000	Cross
Op.	Eastern Oil Processing Co.	Petty Island	2	3,000	Doherty
Op.	Gulf Refg. Co.	Bayonne	1	800	de Florez
Op.	Standard Oil Co. of N.J.	Bayonne, etc.	20	54,050	Tube-and-tank
SD	Standard Oil Co. of N.J.	Bayonne, etc.	6	10,250	Tube-and-tank
Bldg.	Standard Oil Co. of N.J.	Bayonne, etc.	4	17,200	Tube-and-tank
Op.	Tide Water Oil Co.	Bayonne	4	10,700	Tube-and-tank
Op.	Vacuum Oil Co.	Paulsboro	3	3,500	Cross
Op.	Vacuum Oil Co.	Paulsboro	3	3,000	Tube-and-tank
			44	105,500	
	NEW YORK				
Op.	Standard Oil Co. of N.Y.	Brooklyn & L.I. City	6	8,000	Cross
Op.	Standard Oil Co. of N.Y.	Buffalo	2	2,000	Cross
Op.	Vacuum Oil Co.	Olean	2	1,000	Cross
Op.	Vacuum Oil Co.	Olean	1	600	Tube-and-tank
			11	11,600	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	OHIO				
SD	Allegheny-Arrow Oil Co.	Canton	1	800	Dubbs
Bldg.	Gulf Refg. Co.	Cleves	2	6,000	Own
Bldg.	Gulf Refg. Co.	Toledo	2	6,000	Own
Bldg.	The Independent Producers Refg. Co.	Logan	1	750	Dubbs
Op.	National Refg. Co.	Findlay	1	1,000	Own
Op.	National Refg. Co.	Marietta	1	1,000	Own
Op.	The Pure Oil Co.	Heath	2	4,000	Cross
Op.	The Pure Oil Co.	Heath	4	3,000	Gyro
Bldg.	The Pure Oil Co.	Toledo	3	3,000	Gyro
Op.	The Solar Refg. Co.	Lima	2	2,000	Cross
Op.	The Solar Refg. Co.	Lima	15	4,000	Burton
Op.	Standard Oil Co. of Ohio	Cleveland	4	9,600	Tube-and-tank
Op.	Standard Oil Co. of Ohio	Toledo	2	4,800	Tube-and-tank
Op.	The Stellar Refg. Co.	Marne	1	250	Leamon
Op.	Sun Oil Co. 1/	Toledo	2	10,000	Own
			43	56,200	
	OKLAHOMA				
Op.	Anderson-Prichard Refg. Corp.	Cyril	1	3,500	Winkler-Koch
Op.	Barnsdall Refineries, Inc.	Okmulgee	2	4,500	Cross
Op.	Barnsdall Refineries, Inc.	Barnsdall	3	4,000	Cross
Op.	Beckett Co., Inc.	Beckett	1	2,000	Jenkins
Op.	Bell Oil & Gas Co.	Grandfield	1	1,000	Dubbs
Op.	Bilmont Refg. Co.	Garber	1	1,000	Jenkins
Op.	Champlin Refg. Co.	Enid	1	3,500	Winkler-Koch
SD	Champlin Refg. Co.	Enid	1	2,000	Cross
Op.	Continental Oil Co.	Ponca City	2	7,000	Cross
Op.	Continental Oil Co.	Ponca City	6	9,000	Dubbs
Op.	Continental Oil Co.	Sapulpa	6	1,500	Cross
Op.	Deep Rock Oil Corp.	Cushing	5	3,800	Dubbs
Op.	Eason Oil Co.	Enid	2	2,000	Jenkins
Op.	Empire Oil & Refg. Co.	Okmulgee	1	1,000	Doherty
SD	Empire Oil & Refg. Co.	Cushing	2	1,500	Doherty
Op.	Empire Oil & Refg. Co.	Ponca City	3	2,500	Doherty
Op.	Empire Oil & Refg. Co.	Ponca City	2	2,000	Dubbs
Op.	The Globe Oil & Refg.Co.	Blackwell	1	1,000	Cross
Op.	The Globe Oil & Refg.Co.	Blackwell	1	3,000	Winkler-Koch
SD	The Globe Oil & Refg.Co.	Cushing	2	3,500	Jenkins

1/ Estimated.

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	OKLAHOMA (Continued)				
Bldg.	Illinois Oil Co.	Cushing	1	1,500	Donnelly
SD	Imperial Refg. Co.	Ardmore	4	2,000	Dubbs
Op.	Independent Oil & Gas Co. (Phillips)	Okmulgee	1	3,000	Dubbs
Op.	Johnson Oil Refg. Co.	Cleveland	4	3,000	Dubbs
Op.	Marathon Oil Co.	Boynton	6	900	Slagter
Op.	Marathon Oil Co.	Bristow	3	450	Slagter
Op.	Mid-Continent Pet. Corp.[1]	W. Tulsa	75	15,000	Own
Op.	Producers & Refiners Corp.	W. Tulsa	6	3,000	Dubbs
Op.	The Pure Oil Co.	Ardmore	4	2,000	Dubbs
Op.	The Pure Oil Co.	Muskogee	2	4,000	Cross
Op.	The Pure Oil Co.	Muskogee	2	1,000	Gyro
Op.	Rock Island Refg. Co.	Beckett	1	1,800	Winkler-Koch
Op.	Sinclair Refg. Co.	Sand Springs	4	4,000	Cross
Op.	The Texas Co.	W. Tulsa	9	15,000	Holmes-Manley
Op.	The Texas Co.	W. Tulsa	2	2,500	Pressure-coke
Bldg.	The Texas Co.	W. Tulsa	2	2,500	Pressure-coke
Op.	Texas Pacific Coal & Oil Co.	Wynnewood	2	900	Dubbs
Op.	Tidal Refg. Co.	Drumright	2	5,000	Tube-and-tank
Op.	Tidal Refg. Co.	Drumright	5	2,500	Burton
Op.	White Oak Refg. Co.	Allen	1	2,000	Jenkins
Op.	Wilcox Oil & Gas Co.	Bristow	1	1,500	Dubbs
			181	132,850	
	PENNSYLVANIA				
Op.	The Atlantic Refg. Co.	Pittsburgh	2	4,000	Cross
Op.	The Atlantic Refg. Co.	Franklin	6	6,000	Cross
Op.	The Atlantic Refg. Co.	Philadelphia	10	20,000	Cross
Op.	The Atlantic Refg. Co.	Philadelphia	2	2,600	Lewis
SD	Butler County Oil Refg. Co.	Bruin	3	900	Snodgrass
SD	Conewango Refg. Co.	Langdale	2	500	Snodgrass
Bldg.	Emlenton Refg. Co.	Emlenton	1	750	Dubbs
SD	The Freedom Oil Works Co.	Coraopolis	1	500	Dubbs
SD	General Benzol Corp.	Warren	4	1,000	Ormont
Bldg.	Gulf Refg. Co.	Neville Island	1	3,000	Own
Op.	Gulf Refg. Co.	Philadelphia	4	8,000	Own
Op.	Independent Refg. Co.	Oil City	4	1,792	Snodgrass
Op.	Kendall Refg. Co.	Bradford	2	1,000	Dubbs

1/ Estimated.

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	PENNSYLVANIA (Continued)				
SD	Penn. Oil Products Refg. Co.	Eldred	2	1,000	Snodgrass
Op.	The Pennzoil Co.	Rouseville	2	2,000	Dubbs
Bldg.	The Pennzoil Co.	Rouseville	1	1,000	Dubbs
Op.	The Pure Oil Co.	Marcus Hook	4	8,000	Cross
Op.	The Pure Oil Co.	Marcus Hook	3	3,000	Gyro
Op.	Sinclair Refg. Co.	Marcus Hook	20	20,000	Isom
Bldg.	Sinclair Refg. Co.	Marcus Hook	4	17,300	Isom
Op.	Sun Oil Co. 1/	Marcus Hook	10	25,000	Own
Op.	Valvoline Oil Co.	E. Butler	3	780	Snodgrass
SD	Waverly Oil Works Co.	Pittsburgh	1	1,000	Cross
Op.	Waverly Oil Works Co.	Coraopolis	1	2,000	Cross
			93	131,122	
	RHODE ISLAND				
Op.	Standard Oil Co. of N.Y.	E. Providence	3	6,000	Cross
			3	6,000	
	SOUTH CAROLINA				
Op.	Standard Oil Co. of N.J.	Charleston	3	5,250	Tube-and-tank
SD	Standard Oil Co. of N.J.	Charleston	5	8,750	Tube-and-tank
			8	14,000	
	TEXAS				
Op.	American Refg. Properties	Wichita Falls	2	1,800	Dubbs
Op.	American Refg. Properties	Wichita Falls	1	1,000	Cross
Op.	Atlantic-Pacific & Gulf Refg. Co.	Wichita Falls	1	3,000	Jenkins
SD	Burford Oil Co.	Pecos	2	5,000	Jenkins
Op.	Col-Tex Refg. Co.	Colorado	1	3,000	Richmond
Op.	Continental Oil Co.	Wichita Falls	2	2,000	Own
Op.	Cosden Oil Co.	Big Spring	4	6,600	Jenkins
Op.	Crown Oil & Refg. Co.	Pasadena	3	5,000	Holmes-Manley
Op.	Empire Oil & Refg. Co.	Gainesville	2	2,000	Dubbs
Op.	Grayburg Oil Co.	San Antonio	2	900	Dubbs
Op.	Gulf Refg. Co.	Sweetwater	2	4,000	Own
Op.	Gulf Refg. Co.	Ft. Worth	6	4,200	Own
Op.	Gulf Refg. Co.	Port Arthur	47	68,000	Own
SD	Humble Oil & Refg. Co.	Baytown	2	15,000	Cross
Op.	Humble Oil & Refg. Co.	Baytown	20	143,000	Tube-and-tank
Op.	Humble Oil & Refg. Co.	Ingleside	2	15,000	Tube-and-tank
Op.	Humble Oil & Refg. Co.	McCamey	2	15,000	Tube-and-tank

1/ Estimated.

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	TEXAS (Continued)				
Op.	Iowa Park Producing & Refg. Co.	Iowa Park	1	1,200	Jenkins
SD	LaSalle Pet. Co.	Burkburnett	1	1,800	Jenkins
Op.	Magnolia Pet. Co.	Beaumont	20	55,000	Cross
Op.	Magnolia Pet. Co.	Ft. Worth	1	750	Controlled coil
Op.	Magnolia Pet. Co.	Ft. Worth	1	3,000	Cross
Op.	Magnolia Pet. Co.	Luling	1	3,750	Controlled coil
Op.	Marathon Oil Co.	Ft. Worth	9	1,350	Slagter
Op.	Misko Refineries, Inc.	Mirando	1	1,000	Own
SD	Motor Fuel Products Co.	Laredo	1	2,000	Jenkins
Op.	Panhandle Refg. Co.	Wichita Falls	2	1,200	Dubbs
Op.	Pasotex Pet. Co.	El Paso	1	3,400	Richmond
SD	Petroleum Conversion Corp.	Texas City	1	600	True vapor-phase
Op.	Phillips Pet. Co.	Borger	6	7,500	Own
Op.	The Pure Oil Co.	Nederland	26	13,000	Cross
Op.	The Pure Oil Co.	Nederland	3	3,000	Gyro
SD	Richardson Refg. Co.	Big Spring	2	5,000	Jenkins
Op.	Shell Pet. Corp.	Houston	6	12,000	Dubbs
Op.	Simms Oil Co.	W. Dallas	2	3,600	Cross
Op.	Sinclair Refg. Co.	Houston	23	30,400	Isom
SD	Sinclair Refg. Co.	Houston	5	18,000	Isom
Bldg.	Sinclair Refg. Co.	Houston	2	2,600	Isom
SD	Star Refg. & Prod. Co.	Ft. Worth	1	200	Fleming
Op.	Stone Oil Co.	Texas City	1	1,800	Jenkins
Op.	Taxman Refg. Co.	Wichita Falls	1	1,500	Donnelly
Op.	The Texas Co.	Amarillo	1	1,250	de Florez
Op.	The Texas Co.	Amarillo	1	1,500	Pressure-coke
Op.	The Texas Co.	El Paso	1	1,500	Holmes-Manley
Op.	The Texas Co.	San Antonio	2	3,000	Holmes-Manley
Op.	The Texas Co.	Houston	1	1,000	Tube-and-tank
Op.	The Texas Co.	Port Arthur	38	80,000	Holmes-Manley
Op.	The Texas Co.	Port Arthur	1	2,500	de Florez
Op.	The Texas Co.	Port Arthur	2	5,000	Pressure-coke
Op.	The Texas Co.	W. Dallas	2	3,600	Holmes-Manley
Op.	The Texas Co.	W. Dallas	1	3,000	de Florez
Op.	The Texas Co.	W. Dallas	3	4,000	Pressure-coke
Op.	Texas Pacific Coal & Oil Co.	Ft. Worth	1	800	Cross
			273	575,300	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	UTAH				
SD	Utah Oil Refg. Co.	N. Salt Lake City	30	3,800	Burton
Op.	Utah Oil Refg. Co.	N. Salt Lake City	2	6,400	Holmes-Manley
			32	10,200	
	WEST VIRGINIA				
Op.	Carbide & Carbon Chemicals Corp.	S.Charleston	1	800	Gyro
SD	Carbide & Carbon Chemicals Corp.	S.Charleston	1	800	Gyro
Bldg.	Ohio Valley Refg. Co.	St. Marys	1	750	Dubbs
Op.	The Pure Oil Co.	Cabin Creek	3	1,400	Gyro
Op.	Standard Oil Co. of N.J.	Parkersburg	2	2,500	Tube-and-tank
Op.	Tri-State Refg. Co.	Kenova	1	1,800	Jenkins
			9	8,050	
	WYOMING				
Op.	Continental Oil Co.	Glenrock	6	1,988	Burton
SD	Continental Oil Co.	Glenrock	9	1,736	Burton
SD	Egaso Operating Co.	Osage	1	1,000	Cross
SD	Midwest Refg. Co.	Casper	40	6,200	Burton
Op.	Midwest Refg. Co.	Greybull	20	2,740	Burton
SD	Midwest Refg. Co.	Greybull	20	2,740	Burton
Op.	Midwest Refg. Co.	Laramie	13	2,106	Burton
SD	Midwest Refg. Co.	Laramie	7	1,134	Burton
Op.	Producers & Refiners Corp.	Parco	6	3,600	Dubbs
Op.	Standard Oil Co. (Ind.)	Casper	26	9,282	Burton
SD	Standard Oil Co. (Ind.)	Casper	221	38,603	Burton
Op.	The Texas Co.	Casper	4	4,800	Holmes-Manley
SD	The Texas Co.	Casper	1	1,200	Holmes-Manley
Op.	The Texas Co.	Casper	1	1,500	Pressure-coke
Op.	The Texas Co.	Cody	1	1,500	Holmes-Manley
Op.	White Eagle Oil Corp.	Casper	1	1,500	Holmes-Manley
Op.	White Eagle Oil Corp.	Casper	1	2,500	Own
Op.	White Eagle Oil Corp.	Casper	1	1,000	Jenkins
			379	85,129	

I. C. 6510

AUGUST, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

SAFETY STANDARDS AND SAFETY SUGGESTIONS AT IRON MINES IN THE LAKE SUPERIOR REGION

BY

F. S. CRAWFORD

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY STANDARDS AND SAFETY SUGGESTIONS AT IRON MINES IN THE LAKE SUPERIOR REGION[1]

By F. S. Crawford[2]

INTRODUCTION

Although it may be impracticable to adopt standard methods of safe working in all branches of mining, and although it may be impossible to establish a standard for each class of work at each mine, some of the iron-mining companies in the Lake Superior district have found that by adopting certain standard ways of performing certain of their operations they insure both more efficient and safer working conditions. Many companies have adopted the idea that the safe way of doing things is also the most efficient and economical way, and, through the medium of their safety conferences they are trying to find the safest way, and the most efficient way, of performing practically every operation about the mines.

This activity will, undoubtedly, be continued at every operation which uses this method of doing safety work, because in the advance of industry changes must be made in the results required from time to time.

A review of the safety standards and safety suggestions which have been arrived at by some of the companies in this the greatest iron-mining district in the world, will be of value to companies who are interested in improving the safety of their working conditions.

So that attention may be directed to the methods in use, rather than to individual companies, all names have been omitted and companies are referred to only by letter.

COMPANY A

Drifting

In all timbered drifts the company has adopted forepoling as a standard safe practice. This consists of placing two rails about 12 feet long over the caps and blocking the rails on the end. Lagging is placed across the rails in the back of the drift. It is the company's practice to gunite rock drifts. In all main levels a safety zone or shelter hole is cut in the side of the drifts at 100-foot intervals.

Raising

All raises are driven double and cribbed with 5-foot round cribbing timber. One compartment is used as a ladder road and the other for ore or rock. In some cases when driving a raise, one side is driven up first. It is blasted and then cribbed and the back

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6510."

2 - District engineer, U. S. Bureau of Mines Safety Station, Duluth, Minn.

blocked over. Then the other side is driven up. This method is used where the nature of the ground is such that it would be hazardous to work the full width of the raise at one time. Headboards are also used in raises, and in some cases the entire back of the raise is lagged over before drilling. The ground is then drilled through the lagging. Eight-foot ladders are found relatively easy to handle in the subdrifts and to change if they get broken or worn out. All raises are equipped with a raise door and with a timber hoist and timber cradle to handle timber and other supplies. A bar or gate is placed at the bottom of the raise to prevent men from stepping backwards onto the track and thus be in danger of being struck by cars or motors. With this bar in the way a man must turn around and face the track and then he can see the cars or motors approaching.

All chutes are equipped with trolley wire guards. Baffle plates are installed at the mouth of chutes to prevent the trammers or chute tenders from being struck by chunks rolling out of the chute. Since installing this baffle plate the company reports that it has had no accidents to fingers or toes. All chutes are equipped with blow pipes to be used in loading the motor cars. When the blow pipe is found to be ineffective, the ordinary chute bar is used.

BLASTING

Blasting is done with both fuse and electricity. All approaches to the place being blasted must be guarded. Explosive is transported to the working places in bags provided for that purpose. Fuse and detonators are transported in containers. In all working places the explosive, fuse and detonators must be at least 25 feet apart; they are not allowed to be left lying around or on the ground. When making up primers for blasting, a miner is required to take his carbide lamp out of his hat and place it on the side of the drift at least 5 feet away. Lacing the fuse through the explosive is not permitted. Detonators must be inserted either at the end of the stick of explosive or in the side and the fuse must be fastened down with a string. All miners aro furnished with a copper powder punch which is used to make a hole in the stick of explosive to allow insertion of the detonator.

Electric Blasting

When blasting is done electrically, the explosive, or electric detonator, can not be taken into the breast of the drift while the current is on the trolley wire. Holes can not be charged until all machines, drills, and equipment have been removed from the place. A sectionalizing switch must be installed in all places where electricity is used for blasting purposes. Before charging holes, all electric current must be shut off from the drift by opening this section switch. The man in charge of blasting must be stationed at this switch until the blasting is completed. Before charging holes, jumper switches must be in place and a blank igniter placed across the end of the blasting lines. The lead wires for the electric blasting must not be allowed to touch the ground or come in contact with rails or pipe lines. Blasting boxes must be kept locked until ready to blast; one man has charge of the key of the blasting box. All bottom holes must have two exploders. The lead wires must never be wound around spikes to keep them off the ground; insulators must be used, or if it is impossible to use insulators, wooden sprags should be employed to support the lead wires. These rules are strictly enforced in all electric-blasting operations.

Fire Fighting

Apparatus for fire-fighting purposes is as follows: 5 self-contained, mine rescue, breathing apparatus; 4 All-Service gas masks; and 7 self-rescuers, with a goodly supply of oxygen, regenerators, canisters, and other supplies. There is a fire pump underground which is connected to the pump-house sump. In case of fire, the air can be shut off and water introduced into the air lines. In all main-level drifts a 1½-inch fire connection is made in the air lines at every 200 feet. On every shaft station 100 feet of 1½-inch fire hose is kept for fire-fighting purposes only. For fire in the "subs" the regular hose which the miners use for drilling is available. Fire extinguishers are kept underground at all points where they can be readily reached. Two All-Service gas masks are kept in the pump room for emergency.

Standard Safe Practices of Company A

The following is a copy of Standard Safe Practices which are in force at the A mining company:

General

Safety signs indicating the direction of travel to the different exits or shafts must be maintained, and these passageways must be kept open.

All employees, both surface and underground, must be equipped with goggles.

No tobacco cans are allowed to be used as carbide containers, and regular carbide containers must be used to carry the supply of carbide for the shift.

Safety belts must be worn whenever there is danger of a man falling away.

When men are working in the shaft, there must be a sign put up at every station and on surface, reading "DANGER - Men working in Shaft."

Riding on running board of cars or trucks is strictly forbidden.

A safety jenny must be used in erecting all trestles.

An All-Service gas mask should be used when working around chlorine apparatus at the water plant.

There shall be a steel box installed on the landing to keep straw or hay in.

All emery wheels shall be equipped with the Perks Safety Washer.

In steam-shovel work, the shovel operator must blow a warning signal before moving dipper whenever it has been stopped either for cleaning up or when moving ahead.

In welding or cutting it is not permissible to use any open lights around the acetylene tanks.

All empty carbide cans must be washed out with water as soon as they are emptied.

Handling Timber

Timber cradles must be used in all permanent timber raises.

Timber, lagging, or tools must not be piled and left too close to the track.

Timbermen must not stand in bottom of raise while timber, lagging, or tools are being hoisted.

Timber, lagging, or tools must not be thrown down a raise unless someone is stationed at the bottom of the raise or sub to warn anyone that might be passing.

Timber and lagging must not be stored by the shaft.

Timber, lagging, and tools must not be pushed ahead of the motor, but must always be pulled by the motor.

At all raises where timber, lagging, and tools are being hoisted or lowered, there must be a red flashing light or flag used as a warning signal.

All persons handling timber, lagging, or tools must use good leather gloves or mitts.

All permanent timber raises must be equipped with a return bell signal.

Bell signal system must be used in all raises over 20 feet high.

Timbermen must keep the place around the hoisting compartment on every "sub" clean so that no material will be knocked down the raise while handling timber or tools.

Timbermen must make sure that the slings to which the blocks are hung are strong enough to carry the load.

Motor Haulage

Bell on motor must be sounded whenever the motor is started and on going around a curve.

No person shall ride on the cars or motor except motormen and swampers.

No materials to be carried on the motor.

All motor headlights to be kept clean and in good condition, and no motor shall run without headlights.

Motors must stop when men are walking out to the shaft at quitting time.

All motor trains must be equipped with tail bells.

Switching or dumping cars on the fly is forbidden.

Motormen must not run past the block signal safety light.

No one shall run motor unless authorized to do so.

No one shall attach a wire from motor to trolley wire in order to get motor cars out of a drift.

All motor haulage ways must be kept clean.

All chutes must be guarded with a baffle plate in the mouth of chute.

Trolley wire at chutes must be guarded so that it protects half of each car besides the one that is being loaded.

No one shall take dirt out of a chute unless the trolley wire is guarded.

The trolley wire over all pockets must be equipped with the Miami type of trolley guard.

Explosives

Powder must be kept at a safe distance from working place and hung up. Caps and fuse must be 25 feet away from powder.

Fuse and cap containers must be used in transporting fuse and caps from powder house to working place.

Shift boss must supervise all places where they are going to blast through into another place.

No powder boxes or spike kegs are allowed in working places.

Powder boxes that are needed must have the name of powderman written on the inside of box.

All blasting lines must be kept in good order. All blasting lines must be equipped with a single-pole knife switch as a short instead of wire. All current must be cut off trolley wire by a sectional switch before charging holes in a drift. Blasting box must be kept closed--locked.

The lamp must be taken out of the hat when making up the primers and put about 4 feet away.

Copper powder punches must be provided for every gang of miners.

Powder bags, fuse containers, and electric-fuse containers must be used for the transporting of powder and caps from powder house to working place.

No more than 6 inches must be cut from a 6-foot fuse when blasting in any drift or subdrift.

In all bottom holes, there must be two fuses.

All missed holes must be reported to the foreman.

No more than 10 holes to be lighted by one man.

Anyone lighting fuses must have an extra light in case the one he is using goes out.

In case of a missed hole, no one shall be allowed to enter the place for one-half hour.

Hoisting Men

No more than four men and cage tenders shall be allowed to ride on cage while skips are in motion.

Skips must stop when men are being hoisted at end of shift.

No one shall ride with timber, tools, trucks, or powder, excepting cage rider.

Hoist engine man must always give one bell whenever he stops at a level or surface.

A blocking device must be used on cages to prevent the timber truck from moving.

When men are working on skips or cages, a sign reading "Dont start, man working on skips" must be on display on the stand in engine house.

At all level stations a sign must be displayed reading "Do not ring unless cage is here."

Main Drifts

All main drifts must be kept clean and equipped with safety zones at every 100 feet.

All timbered drifts must be equipped with a fire connection every 200 feet and shall not be used for other purposes.

All ladder roads off main drift must be fixed so that a man can not step backwards onto the track.

Carbide Disposal

Carbide must not be dumped along the side of drift or any place where it can be seen.

Dump carbide in cars, pockets, raise, or tunnels where there is no running water.

No carbide shall be dumped on surface.

Mining

Chain and iron wedges are the standard adopted for slusher work.

All slusher hoists must be guarded.

All miners working with slushers must have a pair of good leather mitts or gloves.

All shift bosses and trammer bosses are required to make out a safety report at the end of each shift.

Safety belts must be used wherever there is danger of a man falling.

Signs indicating the direction to different exits must always be kept in good crder.

Penalties for infringement of safety rules are to be administered through the operating heads.

In all drifts where timber is used, forepoles must be used.

In raising, ladders must be put in as soon as there is room for an 8-foot ladder.

In all raises over 100 feet high electric blasting must be used.

In raising, ladder must have a clearance of at least 4 inches.

All persons going underground must wear a hard boiled hat.

Use of Goggles

All underground employees shall be equipped with screen goggles, but in certain places it may be necessary to equip certain employees with Willson goggles on account of the dusty conditions.

Also, employees in the powder house shall be equipped with glass goggles. For blacksmith shop employees (except acetylene welding and grinding) King-Built or equivalent goggles shall be used.

Acetylene welding: Three types of goggles are used for the different kinds of welding. For the oxy-welding: the King-Built spectacle type of goggle or equivalent. For cutting: the Wellworth welding goggles, or equivalent. Also, a Hardy goggle with Universal glass is adapted to all sorts of acetylene welding.

Grinding and chipping in machine shop: King-Built sani-glass or equivalent.

Electric welding: Mask for operators and a shield for helpers, and in connection with this, employees using this equipment are hereby warned that they should also cover their hands with heavy gauntlet gloves and their necks and arms with cloth protection, as the rays of the electricity are very injurious to the skin.

Cleaning flues: King-Built goggle or equivalent.

Landers, chunk breakers, and pitmen: Screen, King-Built, or equivalent goggles.

Nozzle operator on gunite machines: Gas-tight goggles as manufactured by the Mine Safety Appliances Co., or equivalent.

Babbitting: Babbitting mask of approved make.

----------, Safety Engineer.

COMPANY B

The safety standards of Company B are in general made to conform with the requirements of the county mine inspector. There are no printed safety standards; everything is left to the discretion of the foremen and shift bosses.

A hose cart and equipment are available for surface fire fighting. There are no daily fire runs between shifts at this mine underground.

COMPANY C

Company C has not set up any specific safety standards for any of the work, nor has it any printed safety rules. All safety precautions are entirely up to the mining captain. Although no fire run is made between shifts, there is always one pumpman underground.

COMPANY D

As a result of safety meetings, a number of safety suggestions have been made and collected by Company D. A total of 677 recommendations were made by employees at the various properties in 1929. This serves to show the importance of having mine safety committees and holding regular monthly safety meetings of all employees. These safety suggestions are under the following classifications: Safety suggestions; sinking; drifting; miners driving raises; tramming; stope mining; top slicing; and open-pit shovel work.

Safety Suggestions

Clothing.-

1. Hard-boiled hat.
2. Good carbide lamp.
3. Extra carbide in can.
4. Tip cleaner.
5. Matches in waterproof match box.
6. Wire-mesh goggles.
7. Woolen underwear.
8. Woolen shirt.
9. Overalls with jacket tucked in.
10. Wool socks.
11. Boots with the hard toe, knee length if mining.
12. Gloves with leather palm.
13. Slicker if working place is wet.

Change House.-

1. No horseplay around change house.
2. No clothes should be placed in contact with radiators or steam pipes.
3. No open lights allowed in the dry.
4. No accumulation of old clothes in the locker.

5. Place refuse from dinner pails in receptacles furnished for that purpose.
6. Be careful with supply of carbide in the dry.

Going to Working Place.-

1. Do not crowd into the cage.
2. Only one light allowed on the cage.
3. No tools, powder, or caps allowed on the cage with men.
4. Signals to be given by boss or cage rider, only, when raising or lowering men.
5. See that cage door is closed and kept closed until release bell is heard at the station.
6. In carrying tools to working place, keep them off your shoulder, so as to avoid coming in contact with trolley line.
7. When a motor approaches, step in the nearest safety station and do not step out into the drift until you are sure it is clear.
8. Keep traveling roads clean.
9. Keep off ladders when tools are being hoisted, lowered, or carried through raises.
10. Loose or broken ladder rungs in main traveling roads should be reported to shift boss or mining captain at once.
11. Keep eyes open for loose in sides of raise, loose on the ladder rungs, and especially in back of sides at top of raise.
12. Keep all sollars clean.
13. Keep eyes open for loose back over traveling road to working place.
14. Stay with partner or guide until new man becomes familiar with the mine.
15. If light refuses to burn or any other evidences of gas are observed, return to the station immediately.

Sinking

1. Experienced shaft miners should be employed if possible, also a capable hoist man and bucket lander.
2. The miners should provide themselves with hard-boiled hats, hard-toed shoes, wire-mesh goggles, and gloves.
3. Good tools are essential to safe shaft sinking.
4. The landing station and engine room should be well lighted.
5. A nonspinning hoist rope should be used.
6. Clevises or safety hooks must be used on all buckets.
7. A cross-head should be provided with a suitable safety device which will effectually prevent the bucket from lowering in case the cross-head should stick in the runners.
8. No one should ride the skip or bucket unless same is equipped with safety catches and protecting roof.
9. When sinking, bucket must be stopped in the clear near the bottom and rung down when needed by the miners.

10. Every one must stand in the clear at the bottom when bucket or other material is being hoisted or lowered.
11. When tools, timber, or other materials are to be lowered or hoisted in bucket, they shall be securely fastened to the hoisting rope or bail of bucket.
12. During sinking, suitable signals must be provided so that signals can be given from the bottom of the shaft.
13. Standard hoist signals should be used, however small the job, and all men involved should be acquainted with them.
14. All necessary material should be kept near the job, in excess of of immediate need, especially in sinking through surface material.
15. A toe-board should be used around the collar of the shaft to prevent chunks and other material from being kicked down upon the workmen.
16. The workmen should be protected with suitable doors and these doors must be kept closed when bucket is being dumped.
17. While sinking, the collars of the shaft or winze should be guarded with substantial railings not less than 40 inches high, spaced far enough from opening to prevent any person from walking into shaft or being hit by falling chunks.
18. Excavated material should be dumped where it is impossible for it to slide back into the shaft.
19. All blasting in shafts must be done by firing electrically after a depth of 5 feet has been reached.
20. The following safety devices and safety methods should be used to fire electrically safely:
    A. Holes should be connected up in parallel when power is obtained from trolley, and in series when connected to firing machine.
    B. Prevent high resistance in circuit by scraping wire ends clean and making all joints tight.
    C. Do not untwist the wires of the shorted electric blasting cap until ready to connect them to the shot-firing cable.
    D. Test all multiple shot circuits with a galvanometer to insure against failures from high resistance or broken wires.
    E. Use only well-insulated firing cable and keep it dry.
    F. Suspend the firing cable over a dry board in center of shaft keeping it from coming in contact with timber, pipe lines, etc.
    G. Firing cable should be "shorted" on end where the short portable cable is inserted in the circuit.
    H. Use a short cable with proper electrical connections to insert in firing cable to complete the circuit. This cable is in possession of man doing the firing only.
    I. Use a spring knife switch which keeps the circuit open except when pressure is applied when firing.
    J. Switch should be enclosed and padlocked and key in possession of man doing the firing only.

K. A switch should be placed in the trolley line so that power can be cut off from shaft vicinity when ready to charge the holes.

L. In retreating from charged holes, always be certain that each section of the firing cable that you are about to connect to, is "shorted" before making connection, and then open the circuit nearer the charged holes.

21. The portion of the shaft which is not timbered should be well trimmed.
22. While cleaning up a cut after blasting, special care should be taken to see that there are no missed holes.
23. The bucket should not be so full that there is any possibility of chunks rolling off while it is being hoisted.
24. All timber longer than 6 feet must be lowered with a clevis.
25. Timber sets should be carried as close to the bottom of the shaft as found practicable.
26. Great care should be used in blocking shaft sets; a jar might loosen a poorly blocked set and the blocking spill down upon the workmen.
27. All boards, planks, loose rock, and other material must be removed from each set as completed.
28. The timber should also be cleaned and inspected after each blast.

Drifting

1. At the beginning of every shift, as miners walk in the drift, they should look for loose ground in the back and on the sides and if any loose is found, it should be trimmed or propped at once.
2. The back and sides of drift at breast should be trimmed after every blast.
3. If timber in the drift is being advanced, poles should be put from last set to the breast and blocked tightly, so that no loose can fall between the lagging.
4. Where timber is necessary, it should be erected, using staging planks not less than 2 inches thick, and the staging should be held up by safety chain hooks.
5. If muck is to be scraped, no man should stand so as to be hit by ropes or block should the eyebolt break or pull out.
6. As muck pile goes down, the breast and sides should be trimmed, and the miners should be on the lookout for missed holes. The captain and shift bosses should also be on the lookout for missed holes.
7. Never start drilling while any missed holes are present.
8. When using a bar for drilling, be sure that both footings are solid. Soft wood plank should be used to wedge against.
9. Use goggles when starting to drill in hard ground.
10. When blasting, have an extra light near by.
11. When two drifts are within 30 feet of holing through, the men should notify each other as to when they expect to fire.

12. One man should not attempt to light all the fuses alone at a breast of any size.
13. Guard entrance of drift so that no one may enter while charges are exploding.
14. Count all charges as they explode and report any missed holes to the foreman promptly.
15. The back and sides of all timbered drifts should be thoroughly lagged.

Miners Driving Raises

1. Never drill holes for blasting deeper than can be broken clean to the bottom. This should produce the least shattering effect on the face left, and falling chunks are not as prevalent as when the cut is poorly broken.
2. Green hardwood poles, 3 to $4\frac{1}{2}$ inches in diameter, should be used for staging. Stagings should be spaced not over 4 feet apart. Good hitches should be picked for each pole. Cut hitches for stage poles down from the top and not up from the bottom. If ground is very hard, holes can be made with "pop" holes and light blasts. Staging planks should reach from wall to wall on opposite sides, and should be not less than 2 inches thick and should be examined for cracks.
3. The chain grizzly should be hung from poles not over 10 feet from the face of the raise, before drilling the new cut. When raises are proceeding from sublevel to sublevel above, they should be covered with semipermanent grizzlies at the last "sub" holed into.
4. When blasting, a man should be stationed at the bottom of the raise to furnish light or help to the man lighting the holes in case he should lose his light or slip. When holing through, drift above must be guarded and bottom of raise should be guarded during blasting. Raises going over 50 feet between levels or sublevels should be blasted electrically.
5. After blasting it is good practice to blow air into raise to clear it of gases. Always be alert and on the watch for gases in long or poorly ventilated raises. After the blast the back and sides of the raise should be trimmed before anything else is done. Broken or loose stage poles should be replaced at this time.
6. Good tools, such as axes and saws, are essential to safe raising; for the stage, poles and coverings should be cut to an exactness of measurements.
7. Wear goggles when drilling or trimming.
8. The foot of the raise machine should be reasonably sharp so as not to slip on the staging blank.
9. Ladders should be used in raising and as near the head as possible.
10. Raise miners should be especially careful in placing their tools so as not to jar them down when moving around or trimming.
11. Always have a rope for hoisting and lowering tools and staging.

Tramming

1. No one excepting the motorman and brakeman are allowed to ride the motor.
2. No one is allowed to operate a locomotive except the men employed for that purpose, or a man authorized to do so by mining captain.
3. Keep the top of the motor free of material of any kind. No tools or other equipment excepting tools belonging to the motor shall be carried on the motor.
4. Keep lights and bell on motor in perfect condition.
5. No material shall be piled close to the haulage ways. Do not allow timber, rails, plank, etc. to accumulate in the main drifts. Keep the drifts clean.
6. Do not put material of any kind in the safety zones.
7. All switches, frogs, and guard rails on main levels should be blocked so that it is not possible for a man to get his foot caught.
8. The trolley in front of each chute should either have a disconnecting switch or should be guarded with an inverted trough. These guards should be long enough so that a man standing on the coupling at either end of the car is still protected from the wire.
9. The bar used in poking into the chute should be pointed on one end only and the other end looped to form the handle.
10. Trolley pole must not be turned while in motion.
11. Trolley poles must project away from the direction of motion of s locomotive, except in cases where this is impossible and then the locomotive must be operated at slow speed and motorman should hold on to pole.
12. In passing over switches, motor must be stopped before motorman or brakeman leaves the motor to throw the switch.
13. Where practicable, in handling timber trucks with the motor, trucks should be pulled by motor rather than pushed. A long drawbar should be used in this connection.
14. When two or more motors are operating on the same track, suitable block signals should be used.
15. See that there is a light near each chute.
16. Use a bar or shovel in moving chunks on lip of chute or car.
17. Keep powder and caps at least 25 feet apart. Keep powder in bag or box and keep caps in metal containers.
18. Guard all approaches when blasting on grizzlies, in chute or drift.
19. See that all chunks are loaded properly so that they will not roll off of car when car is in motion.
20. Motorman should slow down his train when passing men in drift.
21. Motorman should report to electrician, mining captain or shift boss, any defective condition of the electrical equipment.
22. Brakeman should hold on to car with one hand and use a stick to make a link-and-pin coupling.
23. Coupling should be made where there is plenty of clearance in drift, and where the motorman can see the brakeman. If this can not be done, a third man should be present to give the motorman signals.

24. Trammers should report broken or burned out electric lights along drift to electrician, mining captain, or shift boss.
25. Remove trolley pole when motor is unattended.
26. Trammers especially should wear hard-toed boots.

Stope Mining

1. Inspect back closely for loose and watch for cracks or any other indication of weak bench.
2. Keep bench clean at all times.
3. In rigging up or taking down post and tightening or loosening nuts on arm, always stand so that you are facing the stope.
4. Do not let any part of hose hang over the edge of the bench.
5. All tools not in immediate use should be kept in subdrift well back from stope. Dull and sharp steel should be kept separate and racked.
6. Floor of subdrift should be kept clear of all tools, boards, and refuse to reduce the stumbling hazard.
7. Where mining with auger-type machines, safety belts should always be worn when drilling stope holes or uppers unless otherwise instructed by mining captain or foreman.
8. When drilling slice holes on a bench, drill top holes so as not to break more than 7 feet above the bench.
9. In transporting explosives to working place use canvas bag for dynamite and copper containers for caps and fuse such as are provided for this purpose.
10. Keep caps and dynamite at least 25 feet apart and off the floor of drift.
11. Never lace your fuse when making primer. Use string as illustrated in book of rules.
12. Never use a spike to make the hole in cartridge.
13. Smoking is prohibited when handling explosives.
14. Before charging the holes see that they are well cleaned out by the use of a blowpipe or scraper.
15. When inserting cartridge into the hole, see that the flat end of the cartridge is next to the tamping stick.
16. Cartridge should not be rammed too severely when tamping.
17. When ready to blast, be sure that all men in the stope are notified, and guard all approaches to the stope.
18. When shaking a hole do not recharge it at once but wait a sufficient length of time for it to cool.
19. Count all holes, and in case of a misfire, wait 30 minutes before investigating.
20. All missed holes should be reported to mining captain or shift boss as soon as eiscovered.
21. When coming off shift or moving from one sublevel to another, bar off the opening into the stope in which you were working.

Top Slicing

1. After each blast, pole ahead and make your place safe by carefully trimming the back, breast and sides as far as the dirt will permit. Continue to trim the breast and sides as you scrape the dirt away.
2. When the center of your dirt pile is scraped down, be sure to pick down all chunks lying on the dirt pile to prevent them from rolling down on your hands and feet.
3. When timbering be especially careful of large holes in the back. In order to be safe such a hole should be filled in some manner or securely blocked so that it can not fill unexpectedly and throw an excess weight on the slice timber.

   If the back is poor or not covered, be sure to put boards on top of the poles so as to cover the back tight, leaving no openings for chunks to fall through.
4. Look over the timbered part of your slice every day before you begin your day's work and if any of the sets need a prop, prop it up at once, or if a sprag is broken or knocked out, replace it at once. Also trim the sides or block them.
5. The entrance to your slice should be kept clean so that, if it should become necessary to leave your place in a hurry, you can do so without danger of falling down over any obstacles which would delay you.
6. Always keep grizzly poles over the dirt raise.
7. When timbering, always use your chain hooks to make your staging.
8. Always wear goggles when breaking chunks with a hammer.
9. Make a staging to stand on while drilling; this will prevent you from falling to the ground in case the drill steel should suddenly break off. It is also well to guard yourself with one hand to prevent yourself from falling when the steel breaks.
   (Refers to drilling with jackhammer machine)
10. When blasting, notify the miners above and below you and all others close by. After the holes are lit, guard your entrance so that nobody can enter. Count all the holes as they go off and if you do not hear them all, wait 30 minutes before going into the place.
11. When using axes or saws be careful not to cut yourself with them. When not using axes and saws and other tools put them away in a safe place so that when anyone passes by they will not get injured by them.

Open-Pit Shovel Work

1. New employees should be told of the dangers surrounding their work.
2. Never lose an opportunity of repairing or reporting unsafe conditions or practices.
3. Strains and overexertions are serious. Use judgment in lifting or moving heavy objects.

4. Never throw tools or material to men working on boom, and do not leave tools or material on boom when through repairing.
5. When working on boom, keep in mind the injury to men below by falling of tools and material.
6. When working in high places, wear a life belt properly secured to prevent falling.
7. Pitmen should stay away from bank side as much as possible.
8. Tools should be placed on shelf under running board on track side of shovel.
9. Do not use wedges with mushroomed heads.
10. No lumber, boards or timber with projecting nails must be left lying where the nails can be trod on.
11. Nothing shall be piled closer than 8 feet from the center of the track.
12. Wear goggles when breaking chunks.
13. Watch out for chunks falling off cars or rolling down bank.
14. Chunks should be loaded on the cars properly and banks in vicinity of shovel must be kept trimmed at all times.
15. Pitmen should stay off the shovel, especially when shovel is working.
16. Keep chunks and other material from path used by pitmen when moving track sections ahead.
17. Keep your mind on your work so as to avoid dropping material on feet and hands and being pinched or crushed.
18. See that the coaling plank is in good condition and that it is supported firmly at both ends.
19. Never climb through, over, under, or between ore or stripping cars. Wait until the train passes or walk around the end of the train, giving yourself plenty of room for safety.
20. Do not climb on moving equipment.
21. The bucket should be set down on firm ground before doing any repairing, oiling, or greasing on boom engine.
22. Shut the shovel down before doing any cleaning, oiling, greasing, adjusting, or anything else which would cause you to be near moving machinery.
23. Always replace the guards before working the shovel.
24. Never stand or walk under bucket of shovel when the shovel is in operation.
25. Always be sure that the shovel runner sees you when going into the pit in front of shovel.
26. When you are on top of the bank in front of the shovel, never stand in line with the chain on the boom when the shovel is working, as the chain may break and strike you.
27. When pinching wheels, use a long stick so as to avoid being under the running board or the beam of shovel.
28. The shovel runner must be notified when the brakeman is inside the car breaking chunks.
29. Locomotive firemen must be sure that no pitmen or others are near the locomotive when sweeping gangway.
30. Before opening blow-off valve, the fireman or any others should see that no one is in direct line with the blow-off.

COMPANY E

Company E has no standard methods of doing all operations because local conditions vary the requirements. However, a set of standards has been drawn up at their conferences through which they endeavor to set safety standards for various general classes of work. Those that have been covered are standards for the handling and use of explosives, and safety standards for the operation of scrapers, as follows:

Safety Standards for the Operation of Scrapers

1. Before starting operations the scraper hoist shall be lined up properly and spragged securely.
2. Before making repairs or adjustments on scraper equipment the power must be shut off.
3. Riding scrapers is prohibited.
4. Sufficient positions shall be used in setting up the head block at the breast of the drift so that it will not be necessary to handle the scrapers.
5. Sheaves for head and snatch blocks shall be 6 inches or larger in diameter. They shall be examined daily by the miners and frequently by the shift bosses, to make sure that they are maintained in good condition.
6. When there is a long pull as in transfer drifts or around corners, a device shall be provided for giving signals to the scraper hoist operator.
7. Lights shall be installed over the scraper hoist and an extension cord provided for lighting the drift where necessary.
8. The operator shall not move the scraper at any time until, or unless, he has a definite understanding with his partner.
9. Chains less than 1/2 inch in size shall not be used in the operation of scrapers.
10. When scraper is being operated, standing near the breast of the drift or in a direct line between the scraper hoist and head block is prohibited.
11. When handling wire rope, men shall be required to wear leather gloves.

Standards for the Handling and Use of Explosives

Surface Magazines.-

1. A surface magazine shall be located not less than 300 feet distant from a public highway, shaft, or building, and shall be in a protected place. All brush and inflammable material shall be cleared away for a reasonable distance.
2. The magazine shall be constructed of fireproof and bulletproof material and shall have a steel door equipped with a suitable lock. Adequate ventilation shall be provided. A lightning rod shall be erected at least 18 feet away from the building and shall rise at least 6 feet above the highest point of the building and be grounded in moist earth.

3. Detonators shall not be stored in the same building with dynamite or blasting powder.
4. Magazines must be kept clean at all times, and nothing except explosives shall be stored therein.
5. Boxes of dynamite or cans of powder must not be opened in a magazine.
6. Danger warning signs shall be posted at each magazine.

Transfer of Explosives.-

1. Not more than a 3 days supply of powder shall be taken underground at one time.
2. Powder delivered at a shaft shall be taken underground without unnecessary delay. If this is impossible, it shall be stored in a properly constructed building or covered with a tarpaulin.
3. Explosives shall not be loaded on trucks, skips, cages or cars with men, supplies, or machinery, except such men as may be required to handle them.
4. Explosives shall not be left temporarily at a shaft station, but must be removed immediately to a safe place designated for this purpose.
5. If trucks are used on surface or underground for carrying explosives, they shall be specially constructed for this purpose.
6. Explosives shall not be transported in a car next to an electric locomotive or in any car hauled by a gasoline locomotive.
7. When miners transport explosives, they must be carried in the original containers or in jute bags.

Underground Storage of Explosives.-

1. An underground magazine shall be located in a separate drift or chamber, the walls of which must be fireproof, and shall be located at a safe distance from shafts, pump stations, and main travelled drifts.
2. An underground magazine shall be dry and well ventilated.
3. A powder magazine shall not be directly connected with dry timbering. There shall be a space of at least 10 feet between the magazine and the nearest set of timber.
4. A magazine shall be kept locked when attendant is not on duty.
5. Magazine shall be lighted from the outside if possible. Properly installed electric lights, however, are permissible.
6. Open lights shall never be allowed in a powder magazine.
7. Only one days supply of powder from a magazine shall be given out to each gang, which must be carried in a jute bag.
8. A hardwood mallet and wedge shall be used in opening powder boxes.
9. In mines not using underground magazines, dynamite shall be stored in boxes provided with locks.
10. Care must be taken in placing powder supplies, so as not to endanger main haulage drifts or passageways.
11. Powder supplies must not be placed near live electric wires.

Cutting and Capping Fuses.-

1. All fuses shall be cut and capped, at a suitable place provided for this purpose, by an employee designated for this work.
2. The fuse cutting and capping station shall be located at least 50 feet from a dynamite magazine.
3. Only an approved bench-type fuse cutter and cap crimper shall be used.
4. Capped fuses shall be carried in lined metal containers, and they must be kept in these containers until used.
5. The minimum lengths of fuse shall be as follows: for block holes 3 feet, for drifts 5 feet, for raises 6 feet.
6. Only a standard compound or cap seal shall be used for waterproofing a fuse.

Blasting.-

1. A copper or wood skewer shall be used for punching holes in primers.
2. Open lights must be removed from hats and placed at a safe distance while primers are being prepared.
3. Drill holes shall be thoroughly cleaned out with a scraper or blown out with compressed air before being charged.
4. Charging shall be done only by men experienced in this kind of work.
5. Wooden tamping bars shall be used and the tamping shall be done by pressure only.
6. Tamping material shall be used in all dry holes.
7. Loose powder must be left nowhere except in bags or boxes.
8. Miners about to fire shots shall cause warnings to be given in every direction, and all entrances to the place or places where charges are to be fired, shall be guarded while such firing is going on.
9. The number of shots exploding, except in the case of electric firing, shall be counted by the miner in charge of the blast.
10. In shaft sinking, shots shall be fired by electricity.
11. If a miner is uncertain that all the shots have exploded, no one shall be permitted to enter the place where such charges were fired for a period of 30 minutes.
12. Miners shall not extract explosives from a hole that has missed fire, but a fresh primer shall be inserted above the missed explosive and fired.
13. When tight tamping or stemming is used or when for any reason a new primer can not be inserted, a new hole shall be drilled not closer than 2 feet from the missed hole and shall be pointed at such an angle as to eliminate all danger of detonating the old charge. The new hole shall then be charged and detonated.
14. If misfires occur at the end of the shift, they must be reported by the miner to the shift boss, and he in turn shall make a written report of the same for the boss of the on-coming shift.

15. In blasting timber, fuse must be of sufficient length to permit the man lighting it to get to a safe place. In no case shall fuse be less than 3 feet in length.

COMPANY F

In drifting, Company F drift sets vary from 7-foot legs and caps to 10-foot legs and 8-foot caps, depending upon the nature of the ground and the purpose of the drift. In slicing, the legs vary from 5 to 10 feet in length, and the caps are 8 feet. In sub-levels and tramming drifts the legs are from 7 to 8 feet and the caps also from 7 to 8 feet. In motor drifts 8-foot legs are used with 7 to 8 foot joggle caps. Hitches are dug for posts, which, after being set in place, are fastened with two bridles of 1/2 by 3 inch by 8 foot iron. The bridles are sometimes of heavier material and are fastened to posts with a track spike. Two layers of 2-inch soft wood planking, or one layer of 2-inch hard wood planking, 18 inches wide, is placed on top of bridles to be used as a staging. The cap is first lifted onto the stage and from the stage it is lifted to the top of the post. Two to four men lift the cap, as the miners are instructed to obtain sufficient help to prevent straining backs. The timber sets are kept within 2 to 3 feet of the working face to protect the men against falls of roof. Sprags are used at the top and bottom of drift sets and between the posts and the side of drifts. Where necessary, forepoles are driven, in advance of sets, while working at the face. This method is used in extremely soft ground. In all cases, however, forepoles are used in advance of the last set and the back is tightly lagged over the forepoles. Wide lagging is also used in advance of the last set whenever the ground shows any inclination to slough off. The timber sets are also tightly wedged. Miners make their own wedges. Care is taken to see that drifts are kept free of all storage materials and débris, except what materials are necessary while work is in progress.

Raises are timbered with cribbing. In 2-compartment raises the cribbing is carried on an average within 4 feet of the back. In loose ground the cribbing is carried right up to the back, while in hard rock the cribbing is carried to within 6 or 7 feet of the back. During the process of raising, ladders are installed for the full length of the cribbing, but after installation the ladders are required to extend 3 feet above the sollar or last platform. A complete raise set is comprised of three cribbing sets, making a height of 7 to 8 feet.

In some mines, due to the nature of the ground, it is seldom necessary to crib raises unless a dike or faulty ground is encountered. These raises are generally 5 by 6 feet and are used for but one purpose, -- either as a manway or as an ore chute. When these raises are driven any great distance a parallel raise is made and holed through a 100-foot intervals for safety. This allows the miners, after spitting the fuses, to descend only part way in the raise being driven before finding a place of retreat into the other raise.

The time of blasting varies with the different mines but is usually done from 30 to 10 minutes before going off shift. However, in case of a missed hole or in blasting chunks, and in mines where the smoke clears out quickly, blasting is done at any time during the shift. The miners are supposed to take precautions in counting reports as their holes explode, warn nearby workmen, and guard the approaches to the blast. This precaution is doubly in force when it is expected that a drift or raise will hole through into another opening. Detonators are crimped onto the fuses by the miners in some of the mines, and in other mines crimping is done at a central detonator and fuse magazine using an automatic

crimper. A central explosives magazine is used at all mines except one. At this mine a central storage house is used, but the explosive is taken to the working place by the box and stored in special "powder" containers. No explosive is stored within 50 feet of detonators. Explosive is carried in special bags and no man is allowed to carry explosive in addition to detonaor and fuse. The minimum fuse is 4 feet, but as a rule 6-foot fuse is used. The maximum number of holes fired by one man is set at eight, but regardless of the number of fuses spit at one firing, the company requires that not less than two men be present. No detonator less than No. 6 strength is allowed to be used. In wet ground, waterproof fuse is used, and when necessary, electrical blasting is employed. Care is taken to short-circuit the legs of electric detonators and firing lines during the process of loading. From 10 to 14 holes, depending upon the ground, are required to break a drift round. Wooden tamping sticks and powder skewers are provided by the company and used by the men. Whenever it is suspected from counting the reports that a hole has missed fire, the company requires that one hour must elapse before men return to the face just blasted. In some cases, to get more explosive in the bottom of the hole, the miners "shake" a hole with one-half to one stick of explosive before finally charging it, but before this is done, these holes must be tested for temperature.

For fire protection, fire-hose couplings with fire hose attached are provided at all levels. Fire extinguishers are also kept at all levels and manways. In case of fire, gas masks are used for fire fighting, provided that the air is high enough in oxygen to support the flame of a safety lamp. However, if it is found that the air will not warrant the use of gas masks, then oxygen breathing apparatus is placed in service. Air lines are so arranged that water may be turned into them to reach places which can not be reached by the fire hose. The company does not make a practice, however, of daily fire inspection at either surface or underground workings.

COMPANY G

In Company G mines timber is used in all drifting or crosscutting in iron formation or in iron ore where the opening is larger than 4 feet wide and 7 feet high. In drifting or crosscutting in black slates, grey slates, greywacke or ground hard enough to stand without slabbing off, no timber is used. Where the ground is of a nature that is doubtful, timber is used. The back, sides, and breast are trimmed as soon as miners go back after a blast. Where timber is used, the back is blocked at once, employing four or more 8-foot lagging to block ahead. All blasting is done at noon or when the miners go off shift, so that the air may be freed of gases in their absence and to allow for any delay in blasting due to delayed fire.

All raises in ore or in iron formation are cribbed, except in stoping. Raises in black slate or grey slate are not cribbed if the slates are hard and of a suitable nature. Single cribbed raises are 4 by 3 feet, whereas double-cribbed raises are 3 by 9 feet. A safety chain is used in all raises when they are being driven. After the blast, air is blown for a few minutes, then the miner climbs up, fastens the chain across the opening, and trims the back. The chain is down 15 to 20 feet from the back with a large enough mesh to allow the dirt to go through. Eight-foot ladders are used. This allows the men to carry the ladders close enough to the back so that it is not necessary to climb on the cribbing to get up close enough to trim the back. The ladders are set off to one side and raised 4 inches from the cribbing to allow foothold. Ladders are spliced together by laying a 2 inch plank 2 feet long under the ladder and parallel with it. This allows the dirt to go through under

the rungs of the ladder and does not interfere with footing when climbing. The other side of the ladder is nailed to the side cribbings. All raises are covered, both the mill and ladder road. All manway raises have doors over the top and gates at the bottom, opening inward; the door over the slide is made of metal strips spaced about 1 inch apart to allow for ventilation.

When blasting in drifts, there are two men to light the fuse. An extra carbide lamp is furnished every gang of miners for use when lighting fuses. In subdrifts one man lights while the other is present; in raises one man lights while his partner waits for him at the bottom of the raise.

In case of fire, the air lines are connected with the water lines so that water can be turned into them. Fire doors are placed on each level in each drift. Fire extinguishers are in readiness in the powder house, pump house, and at the stations on each level. All men are instructed so that they know what do do in case of fire. Maps are placed in the dry house and engine shop which show the location of hydrants and accessories, and also give instructions for procedure in case of fire.

COMPANY H

Company H requires headboards in all timbered drifts. No one is allowed at the breast or along the drift while scraping operations are in effect. Special precautions are taken in handling electrical apparatus. In raising, the company has carefully worked out methods for placing headboards and for side protection and has special instructions for the use of safety belts and staging. In blasting, in addition to ordinary safety precautions, the company requires two primers to be used in all bottom holes. The men are limited as to the number of holes they are allowed to light, and an extra man must be present when blasting is being done. Also, special precautions are taken when one opening is holing through into another.

Considerable care is taken in picking and barring down the back. In all subdrifts and raises, where only a very small amount of timber is used, it is the practice to employ a miner to act as a trimmer, it is his duty to trim all loose rock in the ladder roads and the traveled subdrifts. These places are constantly being inspected by the supervising force and by the company mine inspector. At the working face, or breast, and the territory leading directly to the breast, the miners are held responsible for conditions, especially as to loose ground.

In the open stopes it is required that the method of mining be carried on in such a manner that the miners will not be working or traveling out in the open stope, but will be traveling or working under a back or roof which can be easily reached with a pick or bar so that it is possible to keep loose rock trimmed. Long bars must be used to keep the brow of this back trimmed.

Where the sublevel caving system of stoping is employed, a condition exists similar to that in untimbered raises. The stope is started by working up from the timber with a sloping raise for about three cuts. The stope is kept as small as possible, but during this process it is necessary for miners to go up this raise to trim the loose rock and to do the drilling. In this kind of work considerable care must be practiced in trimming the loose rock. When the raise is up three cuts, it is then enlarged into a stope. Good judgment must

be used here so that the back of the stope above the entrance will not be higher than can be reached by the miners in the work of inspecting the back and trimming the loose rock. When the stope becomes large and the back is too high to be reached by the miners, it is necessary to start another manhole to connect with the stope from farther back in the solid. By using such a manhole the miners avoid going into a stope where they can not be sure of the condition of the back. In the sublevel caving system of stoping much care must be exercised with relation to loose rock, especially in faulted ground and ground traversed by dikes.

In timbered main-level drifts and crosscuts, subdrifts, and sublevel crosscuts, the timber is kept right up to the breast. The miners must go beyond the timber, after the blast, to trim the loose rock, which is done with picks and bars. The back must be well trimmed before headboards can be put up. After the headboards are in place, it is fairly often necessary to use blocking between the headboards and the back. It is also necessary to inspect the back after the headboards are in place to see that it has been properly blocked. It is necessary to trim and inspect the sides and breasts of drifts and crosscuts fairly often during the shift.

There is frequently a distance of 1 or 2 feet beyond the timber where headboards can not be used because the back is too low. In this case it is the practice to use longer drills for starting holes so that the miners, when drilling, will not be compelled to go out beyond the timber to complete their cuts.

In sounding the back for loose ground the miners are continually cautioned to be careful not to sound the back directly over their heads, but to sound it ahead of where they are standing.

In all main-level drifting and crosscutting in rock it is the practice either to gunite or to timber. The guniting as the drift or crosscut advances must be done at regular intervals and must be kept as near as practicable to the breast. From the point where the gunite is left off up to the breast it is the duty of the miners to keep the loose material trimmed and make frequent inspections of the back and sides for loose ground. In addition to the miners' inspections, these openings are being inspected for loose ground every day by the supervising force and the mine inspector. When the gunite spalls off, it is immediately re-gunited or timbered.

Provision is made for putting water into the air lines for fire fighting. Fire hose is kept available at a number of places underground. Fire extinguishers are placed at readily accessible points. Men are drilled in the use of fire-fighting equipment underground, and each shaft is equipped with an ethyl mercaptan warning stench. It has not been the custom to make a fire run every day.

I. C. 6511 June, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXVI

BY

FREDERICK W. LEE

INFORMATION CIRCULAR
DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]
No. 26

Compiled by Frederick W. Lee[2]

List of contributing editors of Geophysical Abstracts:

Ayvazoglou, W., U.S. Bureau of Mines, Department of Commerce, Washington, D.C.
Barton, Dr. D. C., Petroleum Building, Houston, Tex.
Belluigi, Dr. Arnaldo, Corso Vittorio Emanuele 178, Parma, Italy.
Bogoiavlensky, Prof. L.,Central Chamber of Weights and Measures,Leningrad,U.S.S.R.
Eckhardt, Dr. E. A., 327 Craft Ave., Pittsburgh, Pa.
Eve, Dr. A. S., McGill University, Montreal, Canada.
Gish, Dr. O. H., Carnegie Institution, Broad Branch Road, Washington, D.C.
Gorsky, Eng. V., Allatini Mines, Ltd., Skoplie B.p. 134, Yugoslavia.
Hartley, Kenneth, 2404 San Jacinto St., Houston, Tex.
Hutchinson, Prof. W. Spencer, Mass. Institute of Technology, Cambridge, Mass.
Jenny, Dr. W. P., Magnolia Petroleum Co., Dallas, Tex.
Karcher, Dr. J. C., Dallas, Tex.
Keys, Dr. D. A., McGill University, Montreal, Canada.
Knappen, Dr. R. S., Gypsy Oil Co., Tulsa, Okla.
Korzujin, Prof. J., National University of Mexico, Mexico, D. F.
Lane, Prof. Alfred C., Tufts College, Boston, Mass.
Lee, Dr. F. W., U.S. Bureau of Mines, Department of Commerce, Washington, D.C.
Leonardon, E. C., 25 Broadway, New York City.
Numerov, Prof. Dr. B. V., Fontanka 34, Leningrad, U.S.S.R.
Petrowsky, A., Wasilly Ostrov, 21 Linia No. 8-A, Leningrad, U.S.S.R.
Roman, Dr. I., 90 Valley Way, West Orange, N. J.
Ruark, Dr. A. E., University of Pittsburgh, Pittsburgh, Pa.
Scholl, Louis A., Box 1805, Houston, Tex.
Shaw, Dr. H., The Science Museum, South Kensington, London, S.W. 7.
Sundberg, Dr.Karl, Swedish American Prospecting Corp.,26 Beaver St.,New York City.
Truemann, O. H., Humble Oil Co., Houston, Tex.
Van Orstrand, Dr. C. E., Interior Building, Washington, D. C.
von Weelden, Dr. A., Shell Petroleum Corp., Dallas, Tex.
Weaver, Paul, Drawer C, Houston, Tex.
Wright, Dr. F. E., Carnegie Institution, Washington, D. C.
Zuschlag, Dr.Theodor, Swedish American Prospecting Corp.,26 Beaver St.,New York City

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2 - Senior physicist, U. S. Bureau of Mines.

(201) ZUR FRAGE NACH DEM ISOSTATISCHEN MASSENAUSGLEICH IN DER ERDRINDE

(CONTRIBUTION TO THE QUESTION OF THE ISOSTATIC COMPENSATION OF MASSES IN THE EARTHCRUST)

By A. Prey

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 2, 1931, pp. 201-225.

To decide the question raised by Hopfner, whether the isostasy is nothing but an illusion produced by the methods of reduction, a complete numerical investigation has been carried out for the case of a nonisostatic earth. Based on the author's development of the heights of the earth in spherical harmonics, the level surface (geoid) and the values of gravity on the surface of the continents and oceans have been computed. It was thereby taken into consideration that the heights were not to be counted from the surface of a normal earth but from the disturbed level surface. The free-air reduction and Bouguer's reduction were applied, as usual, to the values of gravity obtained in this manner.

It was shown that only in a few regions of the earth were the results of such a kind that an isostatic interpretation according to Hopfner was possible. In general the values of gravity on a nonisostatic earth did not agree with the observations. There was an obvious asymmetry in the north-south and in the east-west direction caused by the terms of the first order in the development. Accordingly, the values of gravity in America and Europe, if treated in the same manner, should differ by approximately $100 \cdot 10^{-3}$ cm./sec.^{2}, but this does not correspond to the observations. The absence of this difference seems to be a proof of the existence of a compensation of masses in the earth crust.--Author's abstract.

(202) CONCERNING GRAVIMETRICAL WORK IN CENTRAL ASIA (IN RUSSIAN)

By P. A. Savitsky

Department of Geological Prospecting in Central Asia, Tashkent, Information Bulletin 1, Aug., 1930, pp. 12-14.

A section of applied geophysics has recently been created at the Department of Geological Prospecting in Central Asia for the purpose of a systematic gravimetrical investigation of Central Asia. The investigation is proposed to be carried out in two directions:

1. General gravimetrical survey by pendulums.

2. Gravimetrical survey of single regions by means of gravitational variometers.

The necessity for this work becomes evident, taking into consideration the great importance of the region of Central Asia from the viewpoint of industry.--W. Ayvazoglou.

(203) DETERMINAZIONE GRAVIMETRICA DI INOMOGENEITA PROFONDE INCLUSE IN PIU ESTESE E DIVERSE INOMOGENEITA

(GRAVIMETRICAL DETERMINATION OF HETEROGENEOUS BODIES LYING AT DEPTH AND INCLUDED IN MORE EXTENSIVE BODIES OF VARIOUS HETEROGENEITIES)

By A. Belluigi

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 227-234.

The author examines the gravimetrical influence of an asymmetric anticlinal formation, lying at a depth, which extends in one direction to infinity, remains always in the same position, but has a nucleus or a cavity.

The density of a nucleus should be greater or less than that of the surrounding matter.

The author also suggests the possibility of establishing the reasons for the disturbance by making a graphic-analytical examination of the turning points observed in the profiles.--Author's abstract translated by W. Ayvazoglou.

(204) ÜBER DIE MITTELUNG VON GRADIENTEN UND KRÜMMUNGSWERTEN UND DIE ANWENDUNG EINER UNDULATIONS-METHODE AUF SCHWERKRAFTMESSUNGEN

(CONCERNING THE MEAN OF GRADIENTS AND CURVATURE VALUES, AS WELL AS THE APPLICATION OF THE UNDULATION METHOD TO GRAVITY MEASUREMENTS)

By J. Koenigsberger

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 293-297.

To free the observed values of gradient and curvature from perturbations due to the inhomogeneity of the terrane and to instrumental errors, a set of every n (3 - 5) neighboring stations is tied together by a vector polygon; the resulting vector is divided by n. The center of the set is found like the center of gravity of n equal masses. The regional trend of the gradient of a larger area is found in the same way.

Notwithstanding the nondetermination of potential problems, it is possible to locate in the field the depth of border faces of hidden mass by the method of least undulations. A body in the earth does not have a wholly regular form; irregularities, holes, excrescences, channels, etc., of its border cause undulations in the isogams. The extension of an undulation - that is,

the distance of half maximum values - is independent of the size of the irregularities and is given only by the depth of these irregularities under the surface of the earth, provided that the size of these irregularities is small ($\leqq$0.1) as compared with their depth.

Only the amplitude of undulations varies with the size of the small irregularities. On account of disturbances by nonhomogeneities near the surface of the earth, it would be necessary to make some statistics of the extension of these undulations.--Author's abstract.

(205) INTERESTING TORSION BALANCE

Editorial note

The Petroleum World, London, vol. 28, No. 358, 1931, p. 146.

A modification of the Eötvös torsion balance is exhibited in the applied geophysics collection now on show at the Science Museum, South Kinsington. A description follows:

> The balance consists of a beam suspended by a fine torsion wire, carrying at its extremities two weights at different vertical heights and enclosed in a metal case which can be rotated about a vertical axis. The position of the brass box enclosing the balance arm is indicated by a horizontal circular scale, and the orientation of the arm relative to it is observed by the aid of a mirror fixed on the balance arm, and a telescope.
>
> In order to protect the balance adequately against all external effects which influence its action and equilibrium, it is completely enclosed in a double-walled brass case, the inner and outer walls being 2 mm. and 4 mm. thick, respectively, and separated by an air gap varying between 5 mm. and 10 mm. In this way the balance is protected against radiation, electrical influences, and eddy currents due to variation of temperature, and extreme sensitiveness is obtained.

--W. Ayvazoglou.

(206) A METHOD FOR DETERMINING THE DISTRIBUTION OF SUBTERRANEAN MASSES BY MEANS OF MAGNETIC AND GRAVIMETRIC OBSERVATIONS (IN RUSSIAN)

By G. Gamburtzeff

Journal of Applied Physics, Moscow, vol. 7, No. 2, 1930, pp. 103-105.

In this article Gamburtzeff describes a new method for interpreting magnetic and gravimetric observations by the utilization of optical effects produced by mixing colors. Diagrams serving for calculation of G_z - that is,

of the vertical component of the anomalous value of the force of gravity - are discussed only. These diagrams are designed on transparent paper and different colors are applied to them; the amount of light penetrating through any square should be equal for each diagram. The images of several diagrams are then projected upon a screen so that their centers coincide with the most characteristic points (with respect to the curve ΔG_z) distributed along the line of observation (the latter must also be reproduced on the screen).

As the number of the squares included in the area sought for is known for each of the colored diagrams a certain summarized color can be composed for this area. After the mean color of the outline of the area is established, the finding of the area itself consists in finding on the screen an area having the same color. In order to obtain a summarized color for a certain area, an apparatus for the optical mixing of colors must be used. The form and position of the area can be determined by comparison of the colors. The results can be made exact and verified by means of a multigravimeter.--W. Ayvazoglou.

2. MAGNETIC METHODS

(207) THE GEOGRAPHICAL DISTRIBUTION OF MAGNETIC DISTURBANCE

By W. F. Wallis

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, Md., vol. 36, No. 1, 1931, pp. 15-22.

A discussion at the Department of Terrestrial Magnetism of the magnetic results obtained by the two MacMillan arctic expeditions of 1921-22 and 1923-24, and a comparison of these results with those from several other stations, indicate that during magnetic storms the greatest disturbance occurs in the region of the zone of maximum auroral frequency, and that there is a close correlation between the curves of magnetic activity and auroral frequency when both are plotted in relation to magnetic latitude. A study of the propagation of different types of magnetic disturbance indicates that all types are not propagated with the same velocity. Possible causes of terrestrial magnetic disturbances are discussed.--Author's abstract.

(208) VARIATION OF HORIZONTAL-INTENSITY VARIOMETER SCALE VALUE WITH TEMPERATURE

By George Hartnell

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, Md., vol. 36, No. 1, 1931, pp. 29-32.

In a paper on horizontal-intensity variometers, the author developed the mathematical relations underlying the operation of horizontal-intensity variometers and showed how, in the case of quartz-filament suspension, the scale value may be controlled and compensation for temperature secured by use of suitably placed auxiliary magnets. Further development of the formulas

shows that if too large a filament is used, change of temperature may have an appreciable effect on the scale value.--Author's abstract.

3. SEISMIC METHODS

(209) DIE ERDBEBEN FINLANDS

(FINLAND'S EARTHQUAKES)

By H. Renqvist

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 3/4, 1931, pp. 145-149.

This is an extract from Henrik Renqvist's work (Finnlands Jordskälv, Fennia LIV, No. 1, 113 pp., 13 figures and 16 tables, Helsingfors, 1930) written by E. Tams, in which the latter says that the thorough monographic work on earthquakes in Finland carried out by Renqvist furnishes a very manifold and stimulating contribution to the study of microseismic phenomena connected with earthquakes. The work is not of regional importance only, as in many respects the author discloses new methods; the question of the causes of Finland's earthquakes is examined with special elucidation.--W. Ayvazoglou.

(210) BODENERSCHÜTTERUNGEN, GEOPHYSIK UND INGENIEURGEOLOGIE

(GROUND VIBRATION, GEOPHYSICS, AND ENGINEERING GEOLOGY)

By Walter Kranz

Zeitschrift für Praktische Geologie, Halle (Salle), vol. 39, No. 3, 1931, pp. 38-40.

The author mentions that although there exists an extensive special literature on the use of practical geology for the technique of engineering in earthquake regions, the cooperation of geologists and engineers is not sufficient.

After a brief discussion of the necessity of constructing earthquake-proof buildings, Kranz concludes that, although no technical guarantee can be given against strong dislocations of the earth's crust during earthquakes, most of the damages and losses caused by natural and artificial earthquakes may be prevented by calculation and measurement in technical construction by close cooperation of geophysicists, geologists, and engineers.--W. Ayvazoglou.

(211) MÉTHODE SEISMIQUE DE PROSPECTION DU SOUS-SOL

(SEISMIC METHOD OF PROSPECTING THE SUBSOIL)

Editorial Note

L'Écho des Mines et de la Métallurgie, Paris, vol. 59, No. 3061, 1931, pp. 288-289.

This paper is an extract from Ch. Maurin's work, L'étude du sous-sol par les méthodes géophysiques.

Principles of seismic method are discussed. Acoustic, geothermal, and radioactive methods are mentioned. In conclusion, the author expresses a general desire for the creation, by the cooperation of all engaged in geophysical prospecting, of a geophysical atlas.--W. Ayvazoglou.

(212) NOTE ON THE MECHANISM OF THE NORTH IZU EARTHQUAKE OF NOVEMBER 26, 1930, IN JAPAN

By S. Fujiwhara and T. Takayama

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 2, 1931, pp. 131-137.

This is the preliminary note on the mechanism of a destructive earthquake which occurred on November 26, 1930, over North Izu in Japan. There appeared three or four systems of faults on this occasion, as shown in a figure. At first sight they are too complicated to be explained in any simple way. By a model experiment, however, the authors found that such complicated fault systems can quite naturally be produced by simple horizontal compressive stresses On assuming such stresses as existing in the actual case, the authors have succeeded in explaining the principal facts actually observed.--Author's abstract.

(213) À PROPOS D'UNE ONDE LONGUE DANS LA PREMIÈRE PHASE DE QUELQUES SEISMOGRAMMES

(CONCERNING ONE LONG WAVE APPEARING IN THE FIRST PHASE OF SOME SEISMOGRAMS)

By O. Somville

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 2, 1931, pp. 247-251.

This paper continues an article published by the author in Gerlands Beiträge zur Geophysik, vol. 27, No. 3-4, 1930 (see Geophys. Abs. 21, p. 9).

Since the publication of the first article Somville has obtained new observations on the PL-wave showing the data for the propagation of this wave beyond 1,400 kilometers. These data obtained by new observations are given in a table.

Another table shows the times of propagation for distances up to 2,400 kilometers.--W. Ayvazoglou.

(214) COLLECTION OF ARTICLES ON THE THEORY OF THE SEISMIC METHOD OF GEOLOGICAL PROSPECTING (IN RUSSIAN)

By P. T. Sokolov

Transactions of the Geological and Prospecting Service of the U.S.S.R., Leningrad, No. 17, 1931, 72 pp.

This pamphlet consists of a series of articles on seismic method of prospecting showing the results of Sokolov's work carried out during 1929 and 1930.

The following four problems are discussed:

1. Resolution of the hodograph function into series.

2. The methods for calculating the hodographs of waves caused by explosion.

3. Some suggestions concerning the theory of seismic prospecting.

4. Application of the seismic method to the measurement of the deviation of boreholes.

A definition of a hodograph is given as follows:

Based on the attainments of the present time seismic investigations, two quantities, mutually connected one with another, may be obtained in a simple and accurate way from the experiments. These two quantities are (a) the distance of the epicenter of the earthquake from the point of the outlet of the seismic wave at the surface of the earth, calculated along the great circle of the globe, and (b) the time of propagation of the wave from the epicenter to the outlet. The functional relationship of these two quantities is the "hodograph" and is represented symbolically by $T = T(\triangle)$, in which T is the time and $\triangle$ the distance of the epicenter from the outlet.

In the first article the author examines the general properties of a hodograph and the relationship of its coefficients to the elastic properties of the medium.

The second article deals with the methods of calculating the function of the hodograph. Two fundamental methods are compared: (1) the method of algebraic calculation from the velocities of waves in the environments under investigation, and (2) the method of trigonometric analysis of the optical angles of refraction and complete inner reflexion. The author is inclined to be in favor of the second method by reason of the ease of the calculations, the simplicity of the hodograph formulas and the convenience of interpretation. The author illustrates his conclusions by calculating hodographs for three beds with the boundaries inclined to one another.

In the third article the author discusses the problems of systems for carrying out the survey and of methods for combining the seismic data obtained for various profiles. This problem is considered to be important as, in order to obtain detailed and accurate results it is necessary to unite the single profiles, in some way, into one complete picture, thus introducing instead of seismic profiles one "seismic field."

In the fourth article the author, taking as a basis M. Malamphy's article, A Seismic Method of Determining the Deviation of Dull Holes (see Geophys. Abs. No. 4, p. 10), makes an attempt to justify the theory of the application of the seismic method to the measurement of the deviation of boreholes.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(215) VORFÜHRUNG UND ERLÄUTERUNG EINES NEUEN MESSGERÄTS ZUR DIREKTEN AUSMESSUNG VON WIRBELSTROM FELDERN BEI GEOELEKTRISCHEN UNTERSUCHUNGEN

(PRESENTATION AND EXPLANATION OF A NEW MEASURING APPARATUS FOR THE DIRECT MEASUREMENT OF EDDY-CURRENT FIELDS DURING GEOELECTRICAL INVESTIGATIONS)

By P. Hülsenbeck

Zeitschrift der Deutschen Geologischen Gesellschaft, Berlin, vol. 82, No. 9, 1930, p. 639.

A brief explanation of the new measuring apparatus was given at a meeting of the German Geological Society held on August 9, 1930. Photographic pictures of a series of results of measurements were presented, and it was shown how the deposits of better conductivity could be exactly distinguished, entirely independent of the position of these deposits with regard to the electrodes. Deposits with better conductivity produced eddy-current fields which were caught independently during the measurement. The difference in the pictures in case of different conductivity, depth, and thickness of the deposits was shown. It was explained how the voltage curve produced during a circular measurement was recorded by the measuring apparatus. The instrument is fixed inside of a water-tight box; all its movable parts can be manipulated from the outside so that measurements can be carried out also during a rain. By introducing the compensator into the box all the electrical connections are established automatically. The compensator is also provided with a water-tight tablet in which notes can be written. It was shown how easily the measuring apparatus can be operated.—W. Ayvazoglou.

(216) ÜBER GRÖSSE UND TIEFENWIRKUNG DER KAPAZITIVEN BEEINFLUSSUNG EINES LEITERS DURCH EINE DISHOMOGENITÄT DES UNTERGRÜNDES. EIN REGISTRIERENDES MESSGERÄT

(ON THE VALUE AND DEPTH EFFECT OF THE CAPACITY INFLUENCE UPON A CONDUCTOR CAUSED BY A HETEROGENEITY IN THE SUBSOIL. A RECORDING MEASURING APPARATUS)

By W. Stern

Zeitschrift fur Geophysik, Braunschweig, vol. 7, No. 3/4, 1931, pp. 166-174.

The value and sign of the change of capacity of a linear cylindrical conductor as a function of the distance from the heterogeneous body of electrical conductivity or the dielectric constant of its surroundings is examined in an experimental way. A recording measuring apparatus, developed for this purpose, by which a change of capacity equal to as much as $\pm 10^{-4}$ can be obtained, is described in detail.

It can be shown that the theoretical relationship which so far has formed the basis for the estimation does not satisfy the true conditions. An exact mathematical expression for the law on which the diagrams obtained are based has, of course, not been found so far. The maximum depth effect was established at 1 1/2 length of the conductor, thus can be considered to be about 150 meters.--Author's abstract translated by W. Ayvazoglou.

(217) DIE GLEICHZEITIGE ÜBERTRAGUNG VERSCHIEDENER SIGNALZEICHEN MIT EINEM EINFACHSENDER UND-EMPFÄNGER

(SIMULTANEOUS TRANSMISSION OF DIFFERENT SIGNALS WITH ONE SINGLE-SENDER AND SINGLE-RECEIVER)

By J. N. Hummel and H. Witte

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 3/4, 1931, pp. 175-182.

The authors examine two connections by which a simultaneous transmission and registration by wireless of two different types of signals, independent one from another, is possible.

The article is divided in the following parts:

1. Definition of the problem (Problem-stellung).

2. Scheme of connection for overlapping receiving (Überlagerungsempfang).

3. Scheme of connection for receiving the current impulse (Stromstossempfang).

The scheme of sending and receiving arrangement, as well as the registrations of the signals are shown in figures.---W. Ayvazoglou.

(218) THEORETISCHE GRUNDLAGEN FÜR DIE ERFORSCHUNG DES ERDINNERN MITTELS GLEICHSTROM

(THEORETICAL FOUNDATION FOR INVESTIGATION OF THE INTERIOR OF THE EARTH BY MEANS OF DIRECT CURRENT)

By J. N. Hummel

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 3/4, 1931, pp. 182-190.

Taking into consideration that some methods used in applied geophysics for the discovery of ore deposits and the study of the tectonics of the uppermost layers may also be applied to scientific research of the physics of the body of the earth and especially because it has recently become possible to investigate by geoelectric methods the vertical change of the electrical conductivity at considerable depths, the author expresses the hope that electrical methods, after being developed further, may be applied for obtaining information on the interior of the earth.

In this article Hummel gives formulas for the measurement of the apparent specific resistance, pointing out that in case of investigation of great depths the curvature of the earth must be taken into consideration, and calculates the differences in values from those applied for plane surfaces.--W. Ayvazoglou.

(219) LA MÉTHODE DE LA CARTE DES RÉSISTIVITÉS ET SES APPLICATIONS PRATIQUES

(THE METHOD OF THE GROUND RESISTIVITY MAP AND ITS PRACTICAL APPLICATION)

By C. and M. Schlumberger

Annales des Mines, Paris, vol. 18, No. 9, 1930, pp. 97-124.

This article was presented by the authors at the International Congress of Mining, Metallurgy and Applied Geology held in Liège in June, 1930.

The article was published in the English language in the Canadian Mining and Metallurgical Bulletin 226, February, 1931, pp. 271-297 (see Geophys. Abs. 23, p. 73).---W. Ayvazoglou.

(220) ÜBER DIE BEZIEHUNGEN ZWISCHEN STÖRUNGEN DES KURZWELLENEMPFANGES UND DEN ERDMAGNETISCHEN STÖRUNGEN

(ON THE RELATIONSHIP BETWEEN THE DISTURBANCES OF THE SHORT-WAVE RECEPTION AND THE EARTHMAGNETIC DISTURBANCES)

By H. Mögel

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 3/4, 1931, pp. 207-212.

In this report Mögel shows, based on practical work, by what means of short-wave technique it is now possible to investigate, by making systematic experiments, the changes of the state of the Kennelly-Heaviside layer. After the report, means were discussed for attaining systematic cooperation in collecting results of observations carried out by the Transradio-Empfangsanlage Geltow near Potsdam and the observatory in Potsdam. The following two main questions were discussed in the report:

1. Proof of change of state of Kennelly-Heaviside layer by studying short-wave phenomena in case of normal variations, as well as in case of disturbances.

2. Observation of short-wave reception disturbances in the Transradio's receiving station across the ocean in connection with magnetic disturbances.

Five figures are added.--W. Ayvazoglou.

(221) DALL' UTILIZZAZIONE DEL RAPPORTO DELLE DISTRIBUZIONI DEI CAMPI POTENZIALE E ELETTROMAGNETICO ALLA DETERMINAZIONE DELLE CARATTERISTICHE DI PROFONDITA E POTENZA DEI GIACIMENTI NEI RELIEVI GEOELETTRICI

(ON THE USE OF THE RELATION OF THE DISTRIBUTION BETWEEN THE POTENTIAL AND ELECTROMAGNETIC FIELDS FOR THE DETERMINATION OF THE DEPTH AND POTENTIAL OF MINERAL LAYERS BY ELECTRICAL PROSPECTION)

By A. Belluigi

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 241-254.

In problems where the calculation of the depths of mineral layers is involved by electrical prospection, the first difficulties are encountered in ascertaining the planimetric dimensions of the layers themselves.

Even if it is true that the solutions which one is able to arrive at are not always perfectly correct, it is nevertheless possible to give solutions which are more or less correct.

In paragraphs 1, 2, and 3 the author illustrates the use of the relation of the distribution between potential and electromagnetic fields, especially fields with a nonelliptical structure.

The calculation of the depth can be carried out by the use of Biot-Savart's formula.--Author's abstract.

(222) ELEKTROGEOPHYSIKALISCHE FELDMESSUNGEN MIT NIEDERFREQUENTEM WECHSELSTROM

(ELECTRO-GEOPHYSICAL FIELD MEASUREMENTS WITH LOW FREQUENCY ALTERNATING CURRENT)

By Theo. Diekmann

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 255-285.

The author describes the carrying out in a Devonian schistous region of a quantitative measurement of the distribution of intensity of a magnetic field interconnected with a low-frequency short-circuit earth current. The vertical component (H_z) of the magnetic field, the horizontal component (H_x) parallel to the line connecting the earth electrodes, and (H_y) perpendicular to this line, as well as the direction of the resulting magnetic vectors, were measured.

Diagrams of the distribution of the components resulting from the measurement of the inclination of the resulting vector and of the angle formed by the horizontal component of the magnetic field with the striking of the rock beds are given for a central line perpendicular to the line connecting the electrodes, which crosses the synclinal fold shown in section.

The distribution of the intensity of the magnetic field of the leading-in wire for various positions of the electrodes, as well as for any line running in a mountainous region, is shown in a series of tables. After the elimination of the magnetic field of the leading-in wire and after the introduction of a fundamental system, depending on the topography of the region, the curves of the distribution of intensity are found; they can serve as a basis for the evaluation because the eddy currents induced by the leading-in wire and the secondary currents induced by the primary field can be disregarded in making measurement with low-frequency alternating current.

The diagrams given in Figures 15, 16, and 17 (corresponding to Figures 5, 6, and 7) which show in addition to a magnetic field rotated in the H_x H_y plane, an inverted direction of the vertical component, H_z, illustrate the deformation of the disturbing field caused by the geological structure of the investigated region.-- Author's abstract translated by W. Ayvazoglou.

(223) INTENSITÄTSGRADIENTEN BEI ELEKTRISCHEN AUFSCHLUSSVERFAHREN

(INTENSITY GRADIENTS IN THE ELECTRICAL METHOD OF EXPLORATION)

By Anton Graf

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 286-292.

Formulas for the intensity gradients of the magnetic alternating field artificially established by electrodes and ring senders are calculated, and attention is drawn to the advantage of the measurement of this value.--Author's abstract.

(224) PRINCIPLES OF THE SWEDISH GEOELECTRICAL METHODS

By Karl Sundberg

Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 298-361.

This paper describes the theoretical principles upon which the Swedish geoelectrical methods are based and gives a summary of the most important practical results.

After a short account of the history of geophysical prospecting there follows a theoretical treatment of the electrical properties of the earth's crust, showing that content and composition of impregnated waters determine the electrical conductivity of rocks.

The geoelectrical methods are classified as potential methods, electromagnetic methods, and inductive methods, and the theory of each group of methods is treated. The importance of Lord Kelvin's theory of images for the comprehension of the potential methods is emphasized by several examples. The following two cases referring to inductive methods are fully treated mathematically:

1. Cylindrical conductor in homogeneous field, the axis of the cylinder being parallel to the field.

2. Horizontal layers with infinite horizontal extension. Principle as well as theory for electromagnetic methods constitutes a combination of potential and inductive methods. Numerous results of small-scale investigations illustrate the theoretical conclusions.

The influence of the magnetic properties is briefly treated.

The arrangements for the investigation of the electric and magnetic field are described and, finally, some practical results are briefly referred to, such as the discovery of the Boliden ore and other deposits in northern

Sweden, also deposits at Buchans, New Foundland, at Britannia, Canada, at Questa, U.S.A., and on Billiton, East Indies, as well as results of structural surveys in the Vienna Basin, Germany, and the United States of America.--Author's abstract.

(225) A TWO-DIMENSIONAL BOUNDARY VALUE PROBLEM FOR THE TRANSMISSION OF ALTERNATING CURRENTS THROUGH A SEMI-INFINITE HETEROGENEOUS CONDUCTING MEDIUM

By Herbert P. Evans

The Physical Review, New York, vol. 36, No. 10, 1931, pp. 1579-1589.

In this paper an infinitely long conductor parallel to the surface of a semi-infinite conducting medium is considered. The medium is supposed to consist of an upper stratum of uniform conductivity σ_1, and having elsewhere a uniform conductivity σ_2. The conductor is supposed to carry an alternating current for which the conducting medium forms the return path. The problem under consideration is that of determining the field vectors throughout space, subject to the condition that the frequency is sufficiently low that displacement currents may be neglected. This problem has been solved by Haberland by the case of a sufficiently thin stratum, this limitation arising through the use of approximate boundary conditions. A solution has also been given by Carson for the case of a homogeneous medium. The present treatment utilizes exact boundary conditions and permits the stratum to be of any thickness. The boundary conditions are formulated for a function ϕ which satisfied the wave equation throughout space. By means of the conditions which must hold at the surface of the conductor and at the two faces of the stratum, together with the conditions at infinity, the function ϕ may be expressed in terms of an infinite integral from which the field vectors are derivable. This integral is expanded into a convergent series of simpler integrals which are given physical interpretation.--Author's abstract.

5. RADIOACTIVE METHODS

(226) NEUERE UNTERSUCHUNGEN ÜBER HÖHENSTRAHLUNG

(NEW INVESTIGATIONS ON PENETRATING RADIATION)

By Werner Kolhörster

Zeitschrift für Geophysik, Braunschweig, vol. 7, No. 3/4, 1931, pp. 199-207.

This is a report delivered by Kolhörster at the meeting of the German Geophysical Association, held from September 11 to 14, 1930, in Potsdam, on the results of the research on penetrating radiation carried out during the last three years ending September, 1930.

The following items were discussed:

Instrumental improvements.
Problems of penetrating radiation (Höhenstrahlung)
Intensity measurments at various latitudes.
Intensity measurements at various altitudes.
Aperiodical intensity variations.
Intensity variations.
Periodical intensity variations.

Scientists working on the investigations of penetrating radiation and the results obtained by them are mentioned.--W. Ayvazoglou.

(227) GEGENFELDUNTERSUCHUNGEN UND BEWEGLICHKEITSMESSUNG KLEINER JONEN

(REACTION-FIELD INVESTIGATIONS AND MEASUREMENT OF THE MOBILITY OF SMALL IONS)

By T. Scholz

Beiträge zur Geophysik, Leipzig, vol. 29, No. 2, 1931, pp. 226-238.

In one of his previous publications (Concerning the Method of Counting Ions in the Free-Air; see Geophys. Abs. 26) Yo Itiwara proved the existence of a "Gegenfeld" in the Langevin apparatus. It was desirable to establish whether the Gerdien and Ebert apparatuses also have such a "Gegenfeld." Measurements carried out on this subject at the Meteorological Observatory in Potsdam proved that this "Gegenfeld" existed and caused big error in the determination of the number of small ions. This error could be diminished by using grounded cylinders suspended inside of the condensers.

A strange behavior of the curves obtained from the measurements has been established; this probably may be used for the determination of the mobility of the small ions in an easier and more correct way than it was done previously.--Author's abstract.

(228) ÜBER DIE KONDENSATION AN VERSCHIEDENEN GROSSEN KONDENSATIONSKERNEN UND ÜBER DIE BESTIMMUNG IHRER ANZAHL

(ON THE CONDENSATION AT VARIOUS LARGE CONDENSATION NUCLEI AND ON THE DETERMINATION OF THEIR NUMBER)

By Hilding Köhler

Gerlands Beiträge zur Geophysik, Leipzig, vol. 29, No. 2, 1931, pp. 168-186.

Based on new investigations concerning the atmospheric ionization and the equilibrium of ionization in the atmosphere, the author brings up the question whether the number of nuclei obtained with Aitken's dustcounter is evidently correct. Thermodynamical considerations as well as experiences of the last years prove that this number is questionable.

Some meteorological phenomena are given as a proof of the investigations. These are:

1. A discontinuity of temperature gradient under or in the lower layers of a newly formed cloud.

2. The coexistence of layers of clouds at different altitudes.

3. An abrupt decrease of the atmospheric permeability before the formation of a layer of clouds.

4. The presence of nuclei in fog.

5. A decrease of concentration of chlorine in hoarfrost (Nebelfrost) with the altitude.--Author's abstract.

6. GEOTHERMAL METHODS

(229) GEOTHERMIC MEASUREMENTS NEAR SURFACE

By A. G. R. Whitehouse

The Colliery Guardian, London, vol. 142, No. 3655, 1931, pp. 210-211.

The object of the present observations was to study the relation of surface air temperature to earth temperature at depths up to 30 feet, and to examine the possibility of a more reliable datum for strata-temperature calculations.

The observations were made at Edgbaston in the grounds of the University of Birmingham at a point 453 feet above Ordnance Datum. The earth thermometers at depths of 1, 4, and 7 feet were in gravel, the 30-foot earth thermometer being in soft sandstone.

In Tables I and II are shown: Air and earth temperatures, annual means for the six years, 1924-1929; and annual ranges of temperature of the thermometers, respectively.

From the results of this investigation the author shows that change in mean earth temperature, with increasing depth near the surface, may be distinctly erratic, and that mean air temperature shows no general relationship to mean earth temperature near the surface. The author concludes that, therefore, the only practical datum in this country for calculations of geothermic gradient is the mean air temperature at the surface taken over a long range of years, and that this datum is at the best a very rough one.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(230) ZUR KLÄRUNG TEKTONISCHER EINZELFRAGEN IM RHEINGEBIET MIT HILFE GEOPHYSIKALISCHER METHODEN

(CONTRIBUTION TO THE EXPLANATION OF PARTICULAR TECTONIC QUESTIONS IN THE RHINELAND WITH THE AID OF GEOPHYSICAL METHODS)

By R. Reichenbach

Zeitschrift der Deutschen Geologischen Gesellschaft, Berlin, vol. 82, No. 9, 1930, p. 633.

This is a brief report delivered by Reichenbach during the General Meeting of the German Geological Society held in Coblery from August 6 to 9, 1930.

He describes the results of three geophysical investigations carried out in the Rhineland.

1. Gravimetric measurements carried out in the Oberrhein-talgraben, to the south of Darmstadt, with the purpose of establishing the position of the border-faults and an approximate determination of the depth of the Graben.

2. Seismic measurements carried out in the Niederrheinischen Bucht between Rövenich and Liblar. Based on the results of a seismic profile the depth of the primitive rocks has been established at many places. The greatest depth, about 900 meters, has been reached near Liblar.

3. Radioactive measurements carried out near Hennef on the Sieg established and proved the position of a fault and of two ore lodes, the existence of which was already known.

The author concludes that the use of geophysical investigations may serve not only mining but also pure geology, as various problems may be explained by them.--W. Ayvazoglou.

(231) PLUMBING THE DEPTHS OF THE EARTH

By Kirtley F. Mather

Scientific Monthly, New York, February, 1931, pp. 165-168.

Plumbing the depths of the earth is a task of the greatest practical as well as scientific necessity, as many of the most fundamental questions, including the discovery of most valuable stores of petroleum and ore, can be answered only when the nature of the deeper interior of the earth is known. A brief explanation how seismic waves may serve for the determination of the elasticity and density of the material encountered by them is given.

Methods of geophysical prospecting (seismic, gravitational, magnetic) serving for plumbing the depths of the earth are discussed in general.--W. Ayvazoglou.

(232) GEOPHYSICAL SURVEYS

Editorial Note

Science, Philadelphia, vol. 73, No. 1893, 1931, pp. 383-384.

A report is given of a special exhibition of apparatus and equipment used in geophysical surveys opened in the Science Museum, South Kensington, London.

The exhibition begins with illustrating general magnetic principles, through specimens of William Gilbert's "terrella," or circular loadstone of data about 1600, to the most complicated present-time instruments.

Progress in gravitational and seismic methods of prospecting is shown in two other sections of the exhibition.--W. Ayvazoglou.

(233) GEOPHYSICS APPLIED TO MINING

By James Boyd

The Colorado School of Mines Magazine, Golden, vol. 21, No. 5, 1931, pp. 18 and 38-39.

After a brief introduction in which the author points out the necessity of close cooperation between the geologist and the geophysicist for obtaining successful results from a geophysical survey, the different methods of geophysical prospecting (gravitational, seismic, thermal, radioactive, magnetic, and electrical) and their advantages and disadvantages for various cases are discussed.--W. Ayvazoglou.

(234) PETROLEUM GEOPHYSICISTS NOW HAVE NATIONAL ORGANIZATION

By John F. Weinzierl

The Oil Weekly, Houston, Texas, vol. 61, No. 3, 1931, p. 60.

The Society of Petroleum Geophysicists, of which Dr. Donald C. Barton, Houston, is the president, came into being on May 20, 1930. Its object is to build up a reference library accessible to members and to serve as a place where the developments in instruments and the advancements in interpretive methods are to be discussed and criticized.

In 1930 the society published its first collection of papers, bound under a cover having the following title: The Society of Petroleum Geophysicists Publication of Papers Presented in 1930.--W. Ayvazoglou.

9. NEW BOOKS

(235) Peters, Leo J., and Bardeen, J. The solution of some theoretical problems which arise in electrical methods of geophysical exploration. Bulletin of the University of Wisconsin Engineering Experiment Station Series, No. 71, 85 pp., Madison, 1930. Price, 75 cents. The bulletin is illustrated with figures and graphs.

(236) Williams, Samuel Robinson. Magnetic phenomena. McGraw-Hill Book Co., New York, 1931, XXII and 230 pages. In the first chapter, Magnetomagnetics, dealing with the magnetic properties of substances, are discussed the definitions and laws and the mathematical and physical conceptions pertaining to magnetic properties, as well as the instruments and methods by which these properties are determined. The succeeding chapters on magnetomechanics, magnetoacoustics, magnetoelectrics, magnetothermics, magneto-optics, cosmical magnetism and magnetic theories and facts, discuss the effect of the magnetic properties upon other properties of substances.

10 PATENTS

1. GRAVITATIONAL METHODS

(237) GRAVITY-DETERMINING DEVICE

Richard Hammer, Pittsburgh, Pa.

United States patent 1,796,150 Patented March 10, 1931.

This invention relates to apparatus for determining variations in the force of gravity including a mass, means for mounting the mass for movement toward and away from the earth in response to change in gravity force, means for generating an alternating current of a predetermined high frequency, including means for varying such frequency by the movement of the mass, and means for measuring the frequency as varied.

Claims allowed - 1.

3. SEISMIC METHODS

(238) METHOD OF RECORDING SEISMIC WAVES

Henry Gordon Taylor, Beaumont, Tex.
Assignor to Geophysical Exploration Co. of Beaumont,
Texas, a corporation of Delaware.

United States patent 1,799,398. Patented Apr. 7, 1931.

The invention relates to a method for ascertaining the presence, depth, shape, and disposition of subsurface strata and other geologic structure.

The general object of this invention consists of providing a method by which a reflected wave is more strongly recorded on a seismogram, and by which other waves are partly or wholly eliminated from the seismogram, making it thereby possible to measure with certainty the time required for an artificially created seismic wave to travel from the surface of the earth down to a reflecting discontinuity and back to the surface, from which data the distance to the reflecting discontinuity may be computed.

Claims allowed - 25.

4. ELECTRICAL METHODS

(239) APPARATUS FOR USE IN DISCOVERING AND DETERMINING OREBODIES

Etienne Samuel Bieler and Horace George Isbister Watson,
of Montreal, Quebec, Canada.

United States patent 1,794,663 Patented March 3, 1931.

The principal feature of this invention is the provision of two flat coils of wire permanently fixed at right angles to each other, but which may be tilted and turned in any desired direction so as to obtain the desired information relative to the location of any orebody in the vicinity.

Claims allowed - 3.

INDEX[3]

3 - The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the table of contents, and the third to the page.

I. C. 6512

October, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE CENTRAL-EUREKA MINE, AMADOR COUNTY, CALIF.

BY

James Spiers

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE CENTRAL-EUREKA MINE, AMADOR COUNTY, CALIF.[1]

By James Spiers[2]

INTRODUCTION

This paper describing the mining practice at the Central-Eureka mine is one of a series of similar papers being prepared by the United States Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States. The mining practices described in this paper are illustrative of the methods employed in the exploitation of comparatively low-grade gold ores under the extremely heavy and swelling ground conditions which occur in the Mother Lode belt of California.

The Central-Eureka Mining Co. comprises a group of adjoining mines in Amador County near Sutter Creek. The principal mines of the consolidated group are the Central-Eureka, formerly known as the Summit mine, and the Old Eureka mine, which is also known as the Hayward Quartz mine. The group is situated in the most productive part of the main Mother Lode belt, the 10-mile portion which lies between the towns of Plymouth and Jackson. The three largest producing mines of the entire belt at the present time are the Argonaut, Kennedy, and the Central-Eureka; these lie within a radius of less than 1-1/2 miles.

The grade of the ore in the Central-Eureka mine varies from $4 to $20 per ton. An average of 160 tons per day is mined from narrow veins under exceptionally heavy ground conditions by square-setting and filling. The ore is treated in the company-owned 40-stamp mill by amalgamation in the batteries and on plates, and by concentration on Frue vanners. An average of 85 per cent of the gold recovered is amalgamated and the remaining 15 per cent obtained in the pyrite concentrates. The concentrates represent about 2 per cent of the ore and range in value from $70 to $90 per ton. The sulphide concentrates are cyanided locally under contract.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6512."

2 - Superintendent, Central-Eureka Mining Co., and one of the consulting engineers, U. S. Bureau of Mines.

Electric power for mining and milling is purchased from a power company at a rate of approximately nine-tenths of a cent per kilowatt hour. Most of the timber used in mining is cut on company-owned lands during the winter months.

The Central-Eureka Mining Co. employs a crew of 150 men, including underground and surface labor at both mine and mill.

ACKNOWLEDGMENTS

The author acknowledges the courtesy of Albion S. Howe, manager of the Central-Eureka Mining Co., for permission to write this paper, and the assistance of William O. Vanderburg,[3] who collaborated in the preparation of the data herein presented.

HISTORY

The Summit mine was located in 1855, and 11 years later a shaft was sunk to a depth of 550 feet. About 1,000 tons of commercial ore was taken out, but as the bottom workings offered no encouragement for further sinking the mine was closed down and the machinery dismantled and sold. In 1893 the Summit property was sold for $6,000 and incorporated under the name of the Central-Eureka Mining Co. Since 1893, with the exception of several minor shutdowns, operations have been continuous to the present time.

The Hayward Quartz mine, one of the most famous of the Mother Lode mines, was first opened in 1852. In 1886, when the mine had attained a depth of about 2,000 feet, and after having produced about $16,000,000, it was closed and remained idle for a period of 30 years. In 1916 a new company purchased the property for $500,000 in cash. This company reopened the old shaft and extended it to a depth of 3,500 feet inclined distance (3,212 feet vertical), and drove several new levels without finding commercial ore in payable quantities; work was discontinued in 1921. In 1924 the property was purchased by the Central-Eureka Mining Co. This purchase avoided possible expensive litigation.

The total production of the Central-Eureka Mining Co. from 1896 to the present time has been about $10,000,000.

GEOLOGY

A number of papers describing the geology of the Mother Lode belt and its mines have been published, the most recent of which was written by Adolph Knopf.[4]

The formations with which the veins are associated in the Central-Eureka properties consist principally of slate and meta-andesite, the latter being locally called greenstone (see fig. 1). Above the 1,000-foot elevation

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3 - Associate mining engineer, U. S. Bureau of Mines.

4 - Knopf, Adolph, The Mother Lode System: U. S. Geol. Survey Prof. Paper 157, 1929, 88 pp.

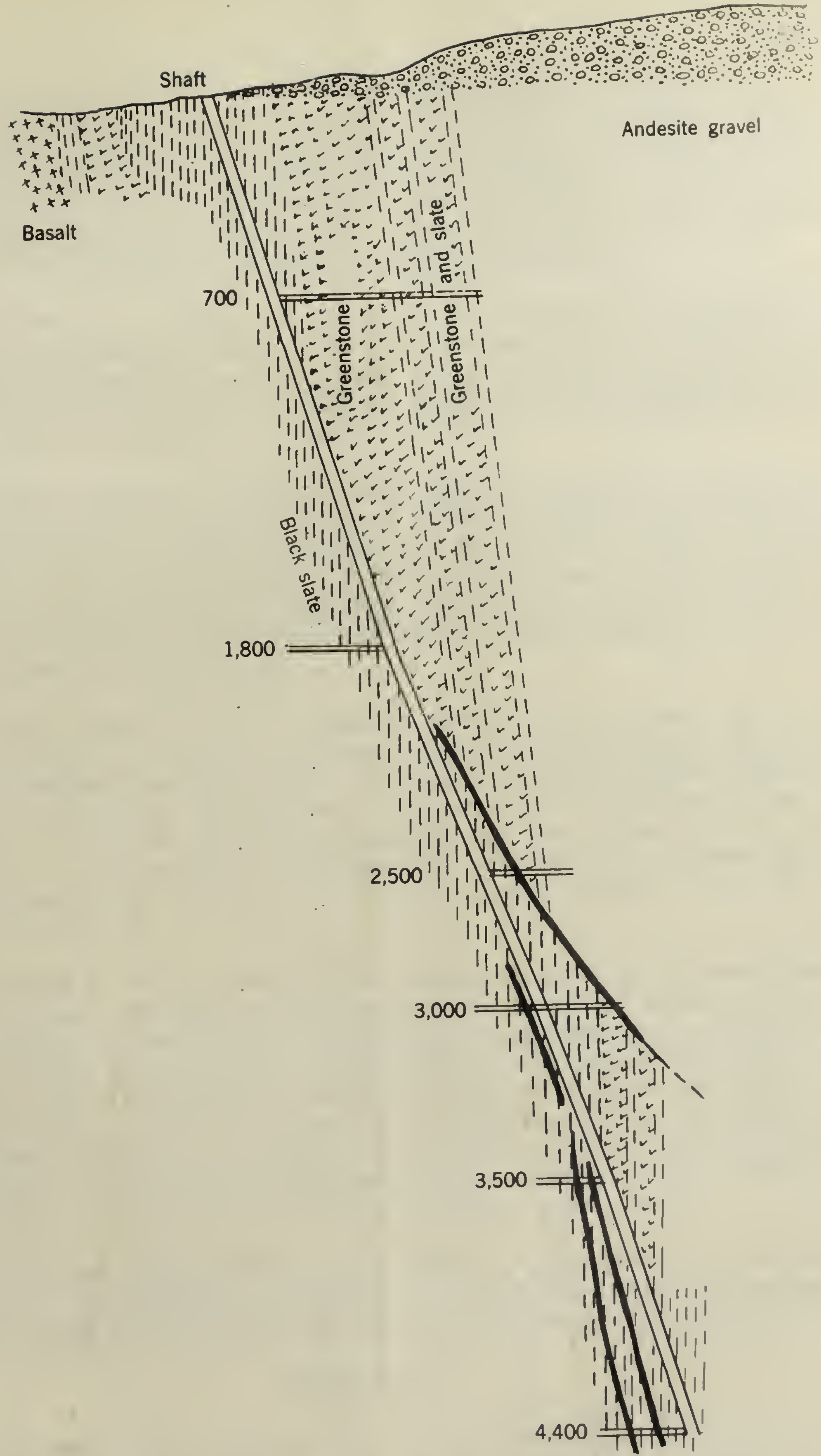

Figure 1.—Section through Central Eureka shaft

of the Central-Eureka mine the so-called "contact vein" was enclosed in slate walls; below the 1,000-foot elevation the hanging-wall slate changed to greenstone. At a depth of about 3,300 feet the "contact vein" became too low-grade to mine profitably. On the 3,500-foot level a new ore shoot was discovered in the footwall of the "contact vein" from which, until recent months, the ore production was taken. The intervening rock between the two veins is slate, which varies in thickness from 30 to 70 feet.

The slates in proximity to the veins are intensely corrugated and their foliation surfaces have taken on a smooth black polish; this structure is so weak that artificial support is required in the mine openings. The greenstones adjacent to the veins have been shattered and subsequently filled with quartz, forming a stockwork of quartz veinlets.

The gold occurs as "free gold" embedded in quartz, or intergrown with sulphides, generally pyrite, but also minor amounts of arsenopyrite and galena. It is commonly present as visible particles or small masses. Not all of the gold is confined to the quartz veins, as ore is sometimes found in the wall rocks. The mineralization in the greenstone constitutes what is locally called "gray ore." Gold values in sufficient quantity to make commercial ore are also found in some places in wall gouges.

The vein filling is mostly quartz. Owing to the intercalation of slate slabs it presents a banded structure, parallel to the walls of the vein. The veins so far worked have invariably been accompanied by gouge ranging from a few inches to 8 feet thick on either or both walls. This gouge causes the ground to swell and necessitates heavy timbering of the stopes. In some places timbers up to 24 inches in diameter are splintered and broken by the tremendous pressure of the moving and swelling ground within a few weeks after being placed in position.

In general it may be stated that the mining method is determined by the following factors: (1) Heavy gouge occurs on either or both walls of the veins; (2) the walls proper are traversed by slickensides and are structurally weak; (3) the vein material, consisting of quartz with intercalated bands of slate, is also structurally weak and crushed to such an extent that in some places spiling must be resorted to for holding the ground; (4) the veins dip from 50 to 80°; (5) the width of the veins ranges from 1 foot to a maximum of 35 feet; the average width mined at the present time is about 5 feet. The length of the ore shoots ranges from 50 to 400 feet.

Deep mining is favored by the low rate of increase of rock temperature with depth. The geothermal gradient at the Central-Eureka mine measured over a vertical range of 4,095 feet is 160 feet per degree Fahrenheit.

EXPLORATION

In the early days, exploration was done by means of surface trenching and shallow shafts. Exploration work is now carried on by means of underground openings: shafts, crosscuts, and drifts. Long haulage crosscuts are not required because the main inclined shaft was originally located to follow the dip of the veins on the hanging-wall side, and the ore shoots have no definite rake.

When drifts encounter changes in formation, crosscuts are driven into the walls to explore for parallel fissures. In the stopes, the crosscuts into both walls for the purpose of providing filling are driven far enough to find any such parallel veins.

Long holes are drilled from the drifts into the walls normal to the dip at 5-foot intervals for testing purposes.

Diamond drilling has been tried but because of the broken-up character of the formation, the low core recovery, and the likelihood of obtaining misleading information due to the manner in which the values are distributed, this method of exploration has not proved satisfactory.

SAMPLING AND ESTIMATING

Hand-sampling is relied upon almost entirely to determine which portions of the ore shoots are ore. The gold values are not uniformly distributed and the quartz is not indicative of the gold present. Some of the best ore mined in the past showed no quartz but occurred in the greenstone and slate.

The faces of horizontal development workings are sampled daily by means of a hand pick, a method which has been found satisfactory as the ground is usually soft and can be picked readily. A face sample is taken by picking channels across the width of the vein or the face, the picked material being caught in one hand and transferred to a canvas sample sack until about a 5-pound sample is obtained. The channels are several inches wide and about an inch deep. If the vein is made up of several layers of quartz showing different characteristics, samples of the individual layers are cut separately and averaged for the full width of the vein. Daily stope samples are taken in the same manner. In raises, where it is difficult to cut a sample from the face, grab samples are taken at the chutes, these being the only chute samples taken in the mine.

Car samples are taken of the ore trammed from the Old Eureka mine to the Central-Eureka shaft, and from the tram cars carrying ore from the shaft to the mill. A mill heads sample is taken from the feeder in the mill. Following the monthly mill clean-up, these mill heads samples are checked against the actual mill returns, with correction for tailing losses.

All assays are recorded serially on loose-leaf forms which show the serial number of the assay, the date of the sample, its location, the character of the rock, as quartz, slate, or gouge, the width of the sample, and the value. These assays are next segregated to similar record sheets for each working place, so that the assay record of any place is readily available. Finally, the assays are recorded on stope floor maps which show the location, character of material, width and assay value of the samples.

Estimates of ore reserves can not be made with a reasonable degree of accuracy because of the manner in which the values are distributed, and the large amount of development work that would be required. Because of the heavy ground conditions and the consequent large outlay necessary for maintenance of mine openings, the ore is not blocked out very far in advance of actual mining operations.

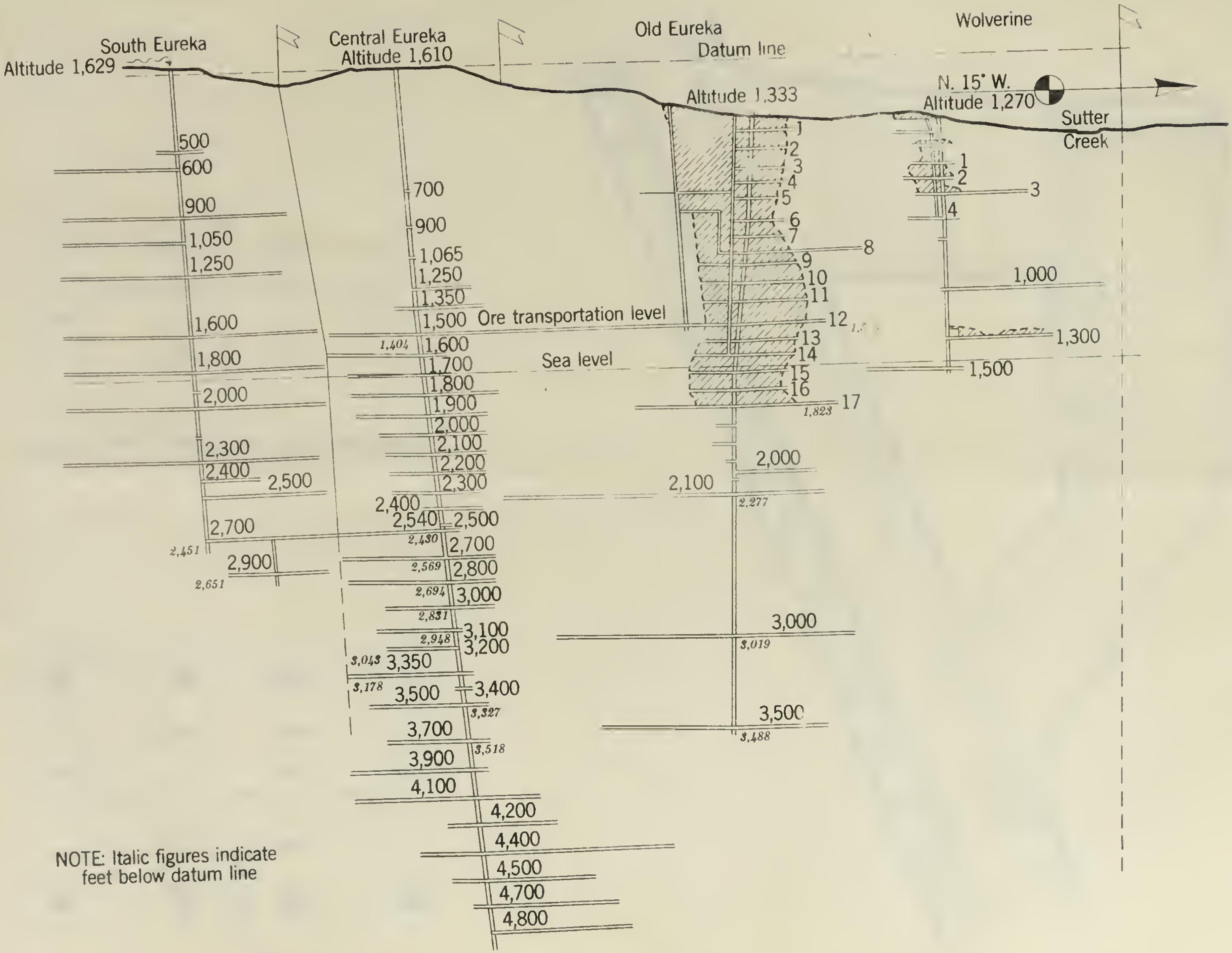

Figure 2.-Longitudinal section of Central Eureka and adjoining mines

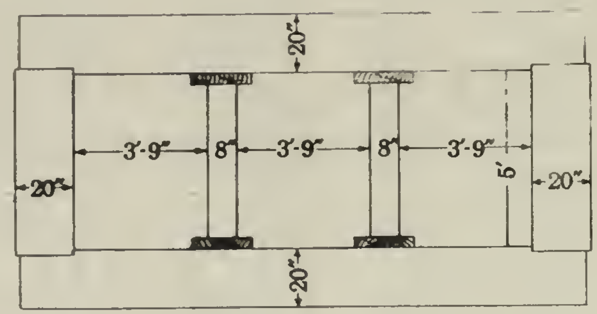

Figure 3.–Main hoisting shaft, timber framing details

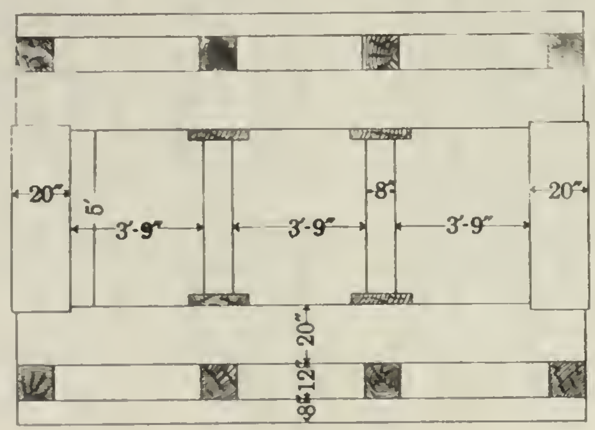

Figure 4.–Jacketed shaft set

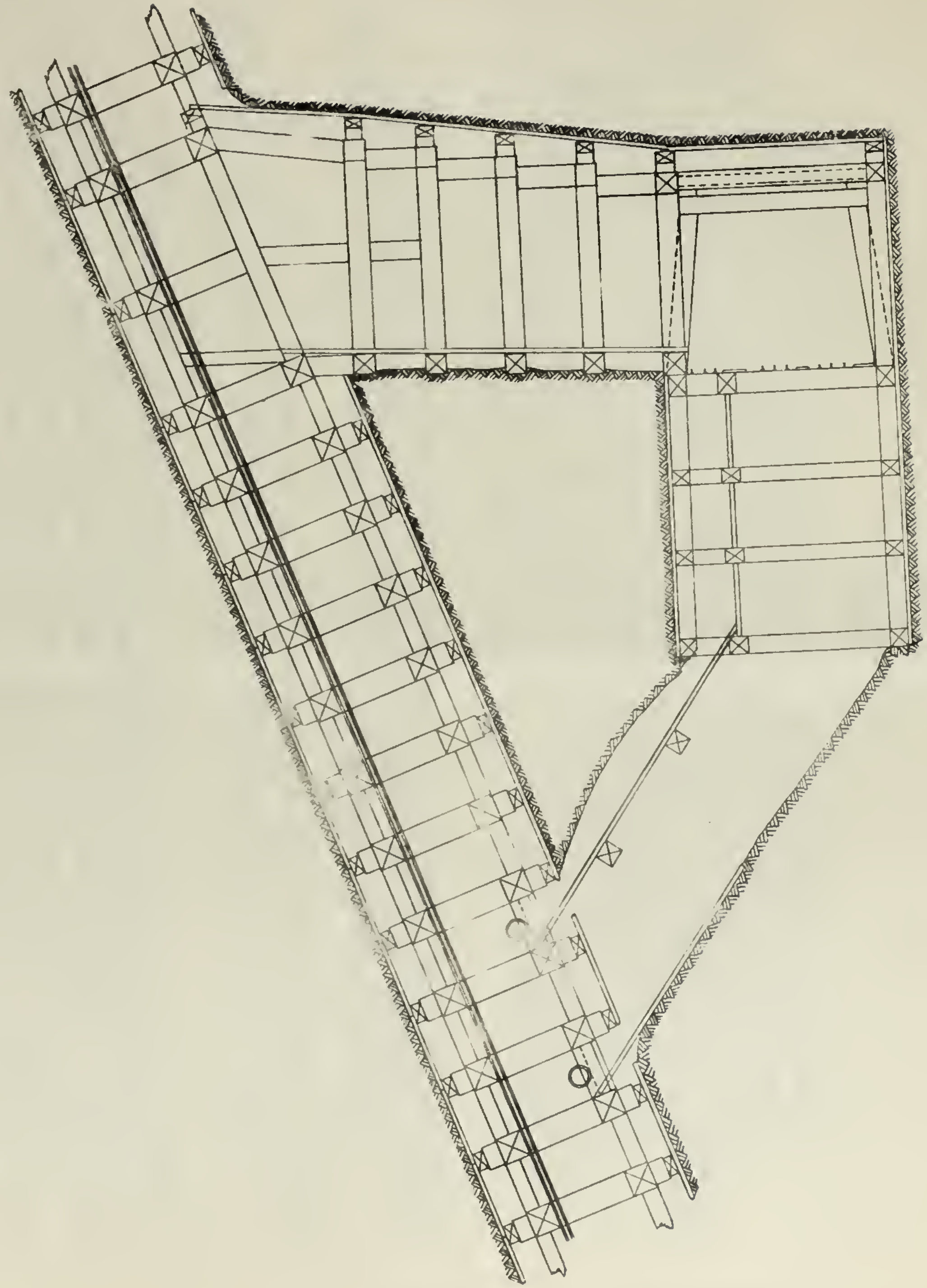

Figure 5.–Station and pocket arrangement

DEVELOPMENT

Shafts

Entry to the mine workings is made by two shafts, the Central-Eureka and the Old Eureka, 4,800 and 3,500 feet in length, respectively, and both inclined at an angle of 70°. In addition the South Eureka shaft situated on an adjoining property, now idle, is kept in repair by the Central-Eureka Mining Co. for ventilation and pumping. This shaft is 2,700 feet deep, inclined distance. All ore mined is hoisted to the surface through the Central-Eureka shaft. At the present time, most of the ore is derived from the Old Eureka property below the 1,200-foot level; it is hoisted to that level and there transported to the main shaft. The level interval varies from 100 to 150 feet vertical distance (fig. 2).

The Central-Eureka shaft is timbered with spruce, which is a long fibre timber and lasts longer than Oregon fir. Wall and end plates are made of logs slabbed on four sides and from 20 to 24 inches in cross section.

The shaft is divided into three compartments, two of which serve as skipways and the other as a manway. Each compartment is 3 feet 9 inches by 5 feet in the clear.

Shaft sets are placed at 5-foot centers. The shaft timber-framing details (fig. 3) are relatively simple compared with other types of shaft framing, as only the wall plates are dapped 2 inches at the ends. The simple style of framing preserves the original strength of the timber without the weakening produced by elaborate joint details, and the ground pressure is sufficient to hold the members of the sets in position without danger of their dropping out. The method of framing has been found to be satisfactory under the heavy ground conditions at the Central-Eureka.

Exceptionally heavy and loose ground was encountered between the 4,500 and 4,800-foot levels of the main shaft and in this area the shaft sets are jacketed on the hanging wall and footwall sides (fig. 4). The jacketed sets are lagged tightly with split lagging spanning two sets to prevent the ground from running. The principal advantage in the use of the jacketed sets is the possibility of making repairs in the shaft or relieving the ground pressure without interfering with hoisting operations.

The inside or floating sets can be kept in alignment by changing the length of the blocking between the sets and jacking the inner set back in position, but the usual method is to keep the blocks of standard length and jack both sets into alignment after relieving the pressure by taking out ground.

The Old Eureka shaft also has three compartments and is timbered with the same type of shaft sets as described. This shaft was idle for 9 years and the bottom portion of it, which has closed completely because of the ground pressure, is now being reopened. The timbers in the upper part of this shaft were originally hand-hewn oak, 20 to 26 inches square, and where these timbers have not broken they are still in an excellent state of preservation after more than 60 years of service.

The station timbering (fig. 5) conforms with standard practice elsewhere except perhaps for one feature. For ease in loading timbers on the trucks the station platform is elevated about a foot above the top of the rails in the haulageway.

Drifts and Crosscuts

On the lower levels the ground is heavy and practically all workings are timbered. Crosscuts from the shaft to the veins and drifts in the veins are timbered with round spruce timber 12 to 16 inches in diameter, and are 5 by 8 feet in section inside the timbers (fig. 6). The sets are placed 5 feet center to center.

When the drift is in ore a bridge is used above the set as shown in Figure 7. The floor of the stope is started from the top of the bridging rather than directly above the drift set timber. This arrangement permits removal of broken drift sets without disturbing the stope fill. The pressure from the walls is usually sufficient to hold the bridge cap and the floor of the stope in place when it is necessary to remove a drift set.

Drifts are driven on the veins. Little difference exists in the ground pressure in the wall rocks as compared with that in the veins, and consequently the expense of maintaining drifts in a vein or parallel to it in either wall would be about the same in both cases. Moreover, the strike of the veins varies to such an extent that difficulty would be experienced in keeping drifts parallel to them. The veins being mined at the present time are usually less than 8 feet in width and the full width of the ore is taken. If the vein is wide the drift is driven along the hanging wall.

The destruction of timber in the Central-Eureka mine is caused almost entirely by heavy pressure; breakage occurs long before the wood is weakened by decay. Numerous slips and fracture zones are encountered which require the drift timbers to be carefully blocked and wedged. In places where swelling ground is encountered bridged sets are used. Sills can not be used under the posts because swelling ground would then lift the track.

All drifts are driven on company account by crews of two men on each of two shifts. These crews on both shifts do all the work necessary to advance the heading.

Raises

Two-compartment ventilation raises are driven from level to level before stoping begins. In addition to their ventilation and exploration value, these raises are later used as starting points for stoping and sometimes for dropping waste fill into the stopes.

Raises are also carried up in the stopes as they advance. These raises may have either one or two compartments. The single compartment raises serve as ore chutes. They are 4 feet by 5 feet in section and cribbed with round timbers 10 to 12 inches in diameter slabbed on two sides (fig. 8). The end plates are placed at right angles to the wall of the vein.

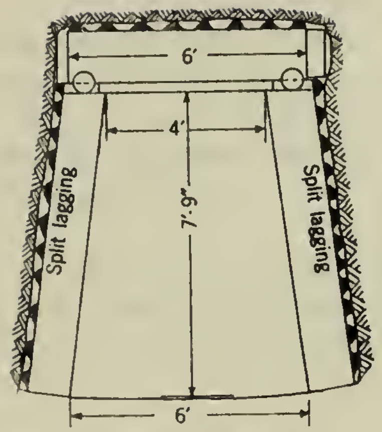

Figure 6.—Drift set

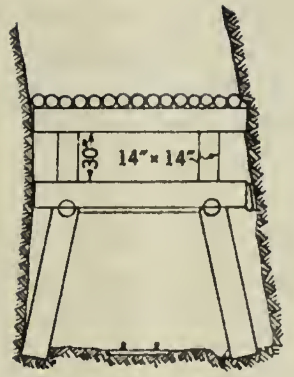

Figure 7.—Drift timbering under stopes

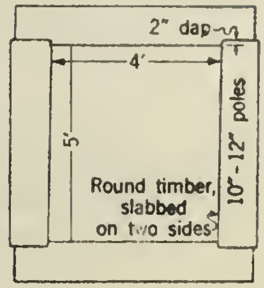

Figure 8.—Cribbed chute

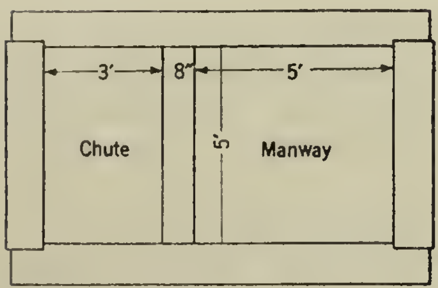

Figure 9.—Cribbed chute, and manway

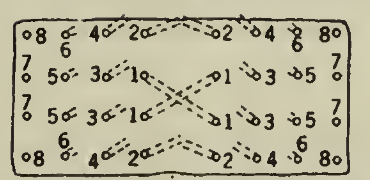
NOTE: Numbers indicate order of firing

Figure 10.—Plan of 32-hole shaft round

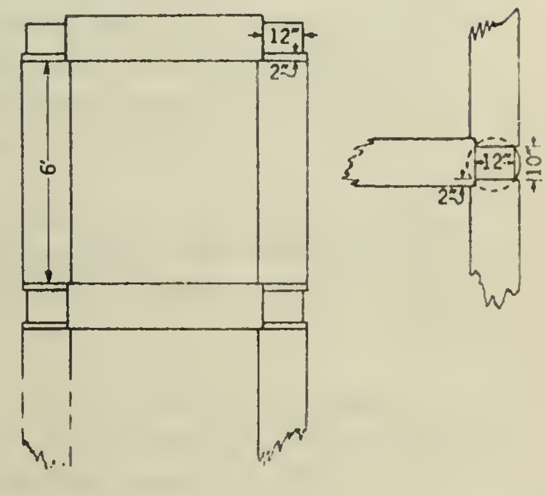

Figure 11.—Square-set for narrow vein

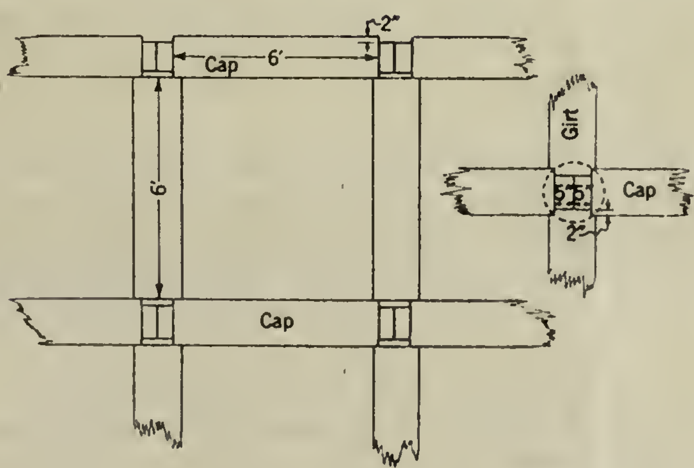

Figure 12.—Square-set for wide vein

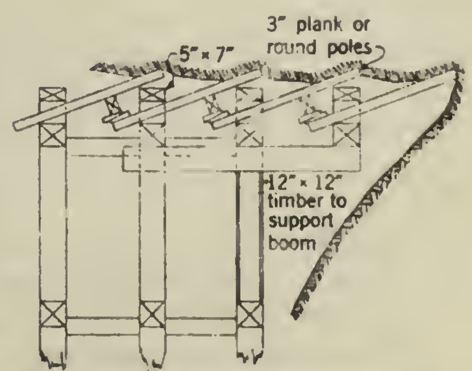

Figure 13.—Method of using spiling in soft or running ground

The two-compartment raises are of similar construction (fig. 9). The chute compartment is 5 by 3 feet and the manway 5 by 5 feet in the clear. The center cribbing is lined with 2-inch plank to prevent small rocks working through into the manway. The chutes are unlined, excepting where sharp bends cause excessive wear on the timber. Chutes are kept full of ore when possible to reduce the wear.

Usually the cribbing extends from wall to wall of the vein. Where the ore is wide the raises are carried up in the square-sets by lagging off two square-sets with split lagging to form a manway and a chute.

Drilling and Blasting

Hand-held jackhammers with 7/8-inch hollow hexagonal steel are used for shaft sinking; Leyner machines with 1-1/4-inch hollow round steel are used in drifting and crosscutting; hand-rotated stoper machines with 1-inch quarter-octagon steel are used in raising and stoping. Standard double-taper cross bits with a change in gage of 1/8-inch for each 1 foot increase in the length of the steel are employed. The maximum length of steel is 7 feet.

The type of round used in shaft sinking is shown in Figure 10. Because of the variable nature of the ground no standard rounds are employed in raises, drifts, or crosscuts. In drifting, the machine is mounted on a 3-1/2-inch column with a cross arm.

Low-freezing gelatin of 40 per cent strength in 1-1/8 by 8 inch cartridges is used in breaking. Electric detonators are used in shaft sinking, and in all other work triple-taped fuse and No. 6 detonators are used.

Air pressure of 100 pounds per square inch at the compressor is used for drilling. Drilling water is taken from behind dams on the upper levels or from the water column in the shaft.

STOPING

Due to the heavy ground conditions practically all ore at the Central-Eureka properties is extracted by overhand stoping using square-sets for support in conjunction with waste filling. Timbers are placed in position as soon as room is available and no opening over two sets (16 feet) in height is made without filling with waste.

Native "mountain pine" round timber is used for square-sets, and ranges from 10 to 16 inches in diameter. This timber is cheaper than the spruce used for timbering permanent openings, but lasts until filling can be placed to take the weight. Two styles of framing the square-sets are used, depending on the minable width of the vein. The style of framing used in veins not over 8 feet in width is shown in Figure 11. The caps range from 6 to 8 feet in length, and the posts are either 6 feet or 7 feet 6 inches in length. Girtwise the sets are spaced on 5-foot centers.

The standard square-set framing for wide veins is shown in Figure 12. The overall length of the caps is 6 feet, of the posts 6 feet or 7 feet

6 inches, and of the girts 4 feet 4 inches. The sets are spaced 5 feet center to center along the vein.

The ventilating raises from level to level are generally used as starting points for each new stope floor. Stope sections are usually 75 feet long, although this figure is variable, and sometimes floors are opened for twice the distance. No pillars are left between sections, and all known ore is removed, excepting shaft pillars.

The sill floor is started from the top of the bridging above the drift timbers. The floor for holding the fill consists of round lagging about 5 inches in diameter. Chutes are spaced at intervals of 25 feet with manways at every third chute, as shown in Figure 14. The chutes and manways are cribbed as described under the section on development, and are carried upward as the mining proceeds. The manways of all two-compartment raises are provided with compressed air hoists using 5/16-inch steel cable. The footwall side of the raise is floored with 2-inch plank for the loads to slide on, and all timber, steel, and drills are thus hoisted into the stopes.

All raises through the stope are kept open until stoping is finished, after which only the raise nearest the shaft is maintained for safety and ventilation purposes.

Chutes are covered with grizzlies of 10-inch round timber spaced 6 inches apart.

Practically all stope drilling is done with hand-rotated stoper drills. A Leyner machine mounted on a column is used on rare occasions, and jackhammers are used to break boulders and sometimes to cut hitches for timbering.

All shoveling in the stopes is done by hand. This controls the spacing of chutes; the customary 25-foot interval requires shoveling the ore not over half that distance. Wooden shoveling floors or sheet-iron plates are laid before blasting.

Waste filling is obtained chiefly by driving chambers in either the hanging wall or the footwall, depending on the firmness of either wall. As stated previously, these openings have an exploratory value, and the strongest point in the wall is selected for them, to avoid caving. They are started as small in cross section as possible, to avoid seriously weakening the stope walls, and are enlarged as they go away from the stope. These waste chambers are usually driven as raises inclined at about 50°. The waste thus obtained is run into the stopes as much as possible by gravity; the final leveling off is done by hand.

Frequently when development work on the level above produces rock of too low a grade for the mill, this material is dropped into the stope through the chute compartment of the ventilation raise and used for stope filling.

Some sorting is done in the stopes when it has been necessary to break down barren or low-grade material from the hanging wall with the ore.

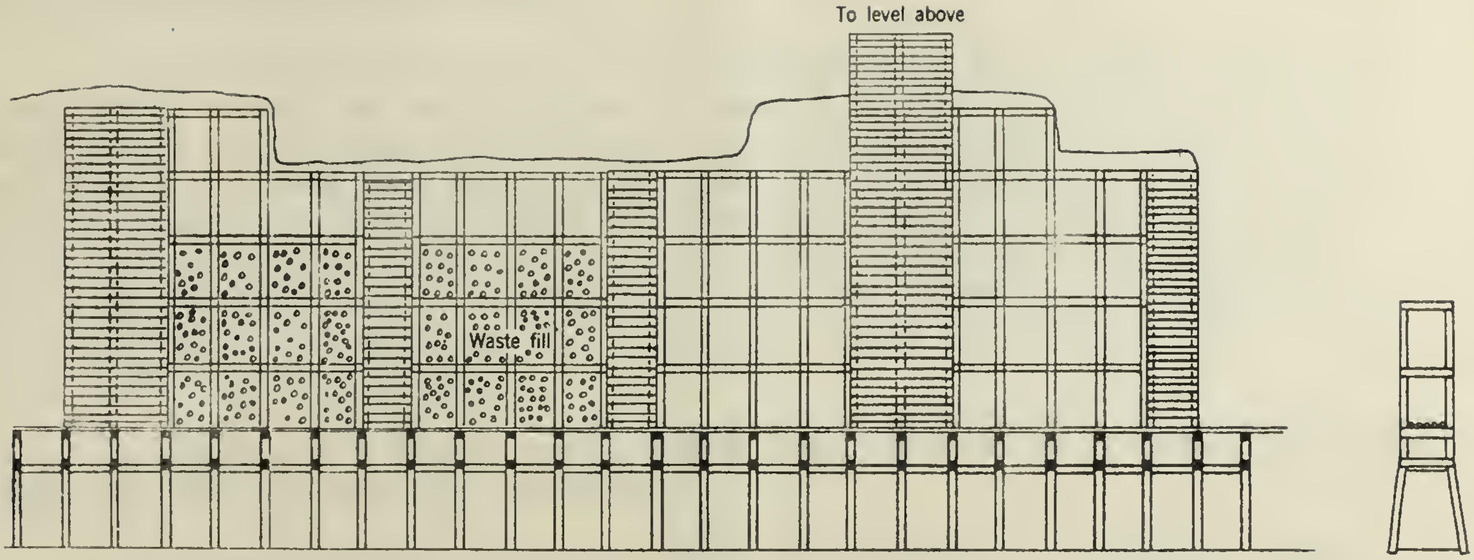

Figure 14.—Narrow stope, longitudinal arrangement

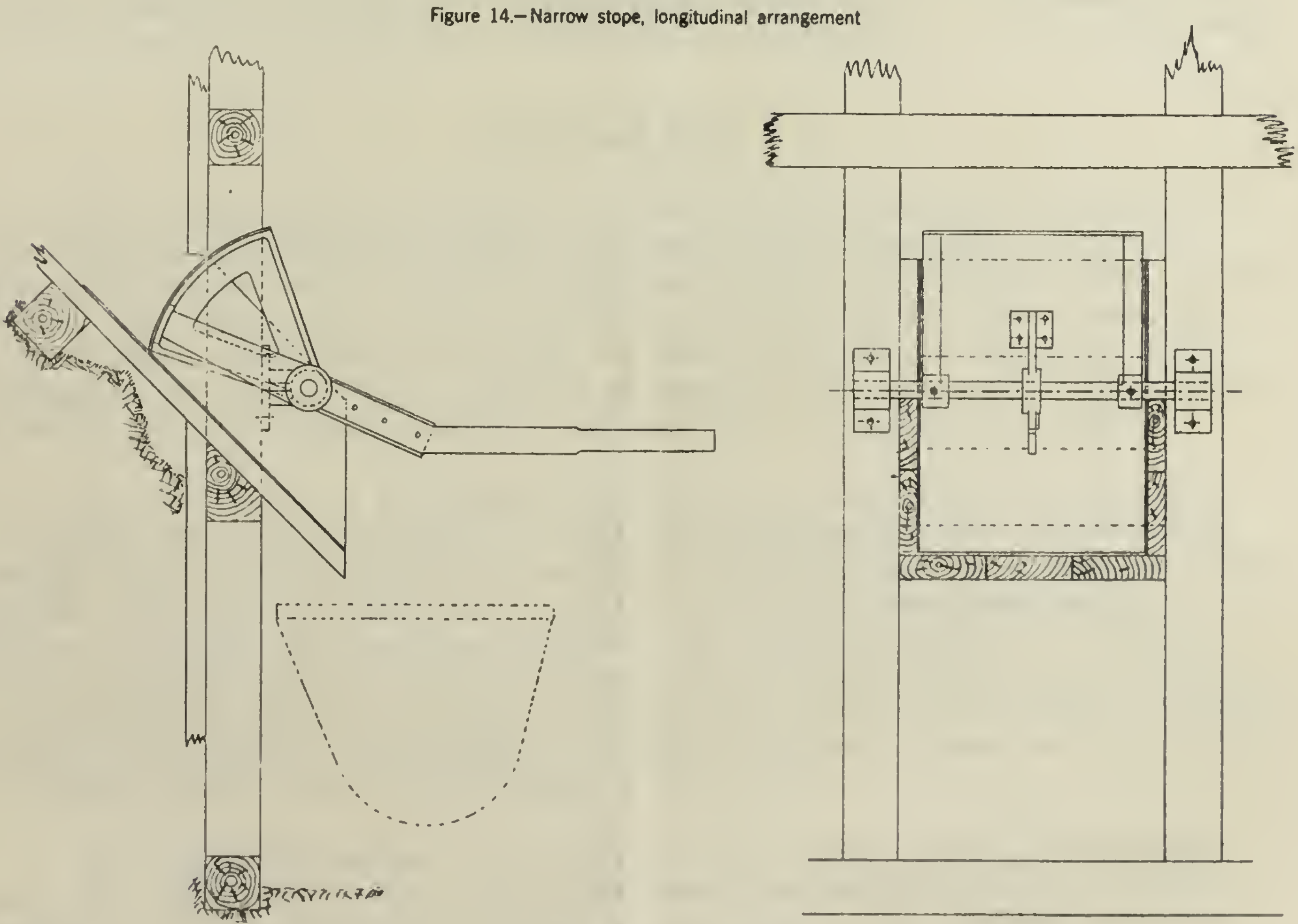

Figure 15.—Loading chute with arc gate

This provides a small amount of waste for filling. No other hand-sorting is done underground or at the surface.

In some cases the ore is soft running ground and in this event its extraction is a more serious problem. A system of spiling is employed in conjunction with the square-set timbering. Spiling consisting of 3-inch plank or round poles is driven over the caps by means of hand sledges and are held in position against the back by booms, as shown in Figure 13. The ends of the spiling timber are not sharpened; the spiling is worked into the ground by driving with sledges and by picking the loose ground away from before the ends. In some places side spiling and breast boards are also required to prevent the ground from running.

A miner and a helper constitute a stope crew at each face. These men do both the drilling and the timbering.

To prevent "high-grading," guards or watchmen are placed in some rich stopes, and these stopes are worked only on day shift. No change-house system or precautions are observed, other than supervision by the change-house man.

TRANSPORTATION

The ore is drawn from chutes of the ordinary timber and plank construction into cars of 16-cubic feet or about 0.8-ton capacity, and hand-trammed to the shaft bins. Cars are equipped with roller bearings and alemite connections for greasing, and are regularly inspected and greased every two weeks. All track underground for hand tramming consists of 16-pound rails with a gage of 18 inches; it is laid to a grade of 0.4 per cent. Ties are of untreated Oregon fir 4 by 6 inches in section and 30 inches in length.

The ore from the Old Eureka workings is transported to the main shaft underground on the 1,200-foot level by a storage-battery locomotive. The locomotive is rated at 3.8 hp., weighs 1-1/2 tons, and is driven by spur-gear drive. The starting draw-bar pull is 600 pounds; when running at a speed of 4-1/2 miles per hour the draw-bar pull is 400 pounds. A train consists of six side-dumping rocker-type cars each having a capacity of 1 ton. These cars have 12-inch diameter chilled cast-iron wheels with roller bearings and alemite connections for greasing. Figure 15 shows a steel arc gate used at the loading chute where these trains are loaded.

The track for mechanical haulage has 25-pound rails; the maximum curvature is a 15-foot radius curve. The total tramming distance from the Old Eureka to the Central Eureka shaft is 1,800 feet; the average cost when hauling 150 tons per shift is about 3.8 cents per ton.

The skips have a nomical capacity of 3 tons each and are 9 feet long, 25 inches wide, and 3 feet deep over-all. Ten-inch steel wheels are used on the skips and the axle is housed in a continuous box filled with grease.

Track for the skips consists of 52-pound rails laid to a gage of 27 inches. It is laid directly on the footwall plates of the shaft. At intervals of 100 feet carriers are used to support the rails. The method of clamping the carriers to the wall plates and attaching them to the rails is shown in Figure 16.

For signalling the hoist engineer two bare copper wires are strung on the end plates and on the dividers about 4 inches apart. The wires carry a current of 24 volts and the bells are rung by short-circuiting the wires with a short piece of steel held in the hand.

WAGE SCALE

Most of the work underground is done on company account. The labor turnover from month to month is generally quite high. The wage scale effective at the Central-Eureka Mining Co. in November was as follows:

Hoist engineer........	$ 5.50
Surface labor.........	3.75
Shaft boss...........	5.25
Shaftman..............	5.00
Miner.................	4.50
Miner helper..........	3.75
Timberman.............	4.50
Timberman helper......	3.75
Trammer...............	3.75
Shoveler..............	3.50
Tool nipper...........	4.50
Pumpman...............	5.00
Skip tender...........	4.75
Motorman..............	5.00

VENTILATION AND SAFETY

Mechanical ventilation is employed to ventilate the deeper portions of the mine. On the 2,700-foot level of the South-Eureka shaft which is on an adjoining property two fans are installed. The larger fan has a capacity of 23,000 cubic feet per minute under a water-gage pressure of 8 inches. This fan is used as a suction fan, and is of the nonoverloading type. The other fan has a capacity of 15,000 cubic feet per minute and operates as a pressure fan against a pressure of 4 inches water gage. The smaller fan is used to reverse the air circuit in case of necessity. The fans are direct-connected respectively to 50 and 25 hp. alternating-current motors.

Compressed air is used to a great extent to ventilate dead ends. To obtain more efficient use of the compressed air and to reduce the amount used a cast-iron blower attachment (fig. 17) is connected to the compressed-air line.

A first-aid man employed by the company gives training in first-aid to picked men once each month. Five sets of Burrell breathing apparatus are kept in condition for immediate use at the mine; training in the use of

this apparatus is renewed once each month. In addition a cooperative station with a full complement of rescue apparatus is maintained at the Central-Eureka property.

Drinking water for mine use is taken from a spring on the 700-foot level of the Central-Eureka mine; the water carried to each shaft station below the 700-foot level in 3/4-inch pipe lines. The type of homemade drinking fountain connected to the line at each level is shown in Figure 18. The pressure in the pipe line is reduced at each level to prevent bursting of the pipe.

Table 1. - Details of costs in units of labor, power, and supplies

Old Eureka mine — Period: Month of September, 1930
Ore mines: 4,143 tons — Mining method: Square-sets with waste fill

Labor (man-hours per ton)	
Superintendence	0.011
Shaftmen (repair)	.343
Miners and timbermen	1.483
Miners and timbermen helpers	.993
Trammers	.375
Shovelers	.270
Skiptenders and hoist engineers	.361
Miscellaneous underground	.058
Topman	.175
Motorman	.050
Surface labor 1/	.419
Total	4.538
Average tons per man per shift (all men chargeable to underground operation)	1.76
Explosives (pounds per ton, 40 per cent strength, low-freezing gelatin)	1.01
Fuse (feet per ton)	3.80
Caps (number per ton)	6.58
Timber (sawed stock, board feet per ton)	7.70
Logs (board feet per ton) 2/	18.08
Total (board feet per ton)	25.78
Power (kw. h. per ton)	86.0
Other supplies in percentage of total power and supplies	17.4
Labor, percentage of total cost	69.3
Power and supplies, percentage of total cost	30.7

1/ Includes electrician, blacksmiths, timberframers, master mechanics and watchman.
2/ Round timber reduced to board feet by Scribner's rule.

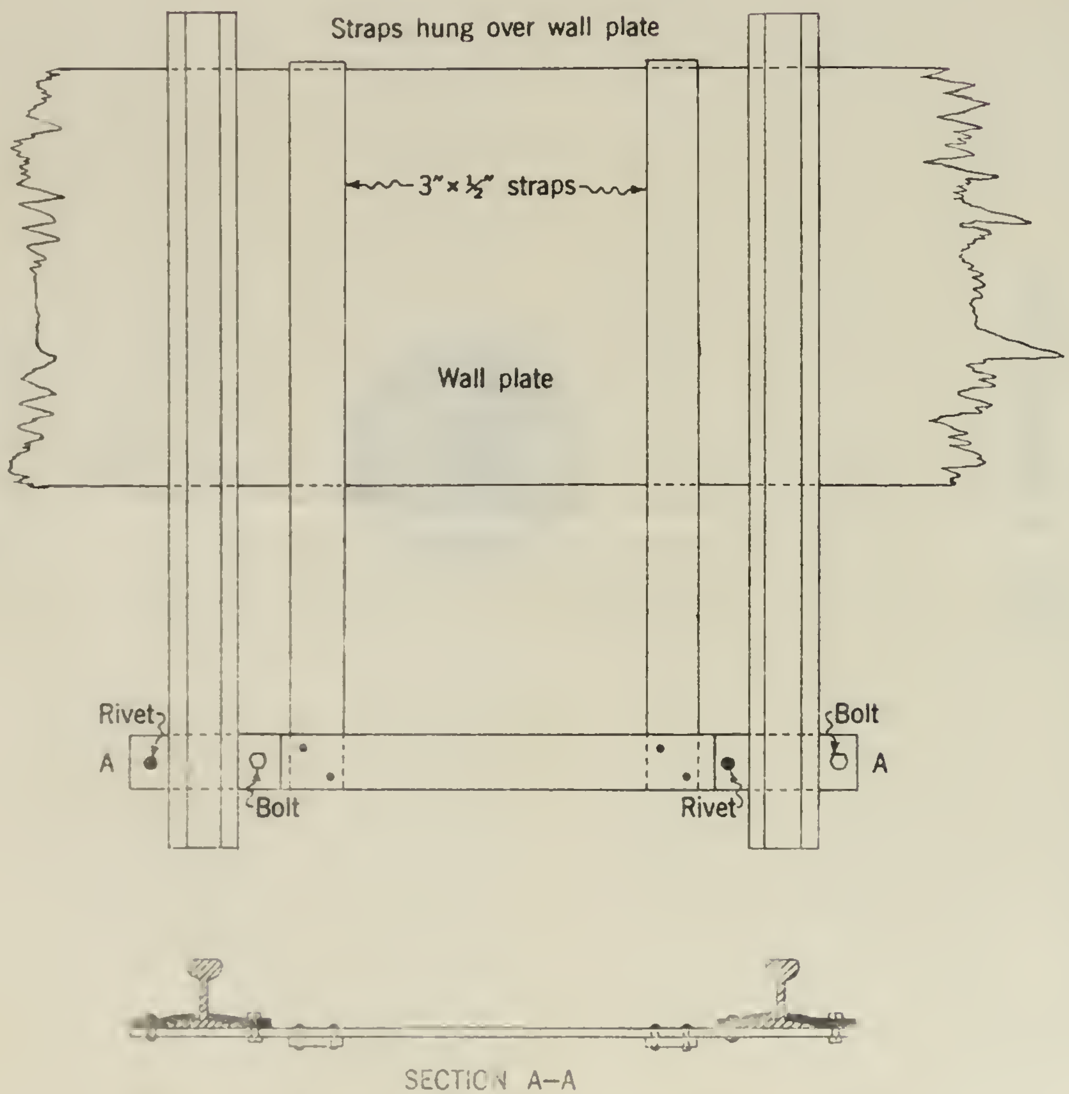

Figure 16.—Rail carrier for inclined shaft

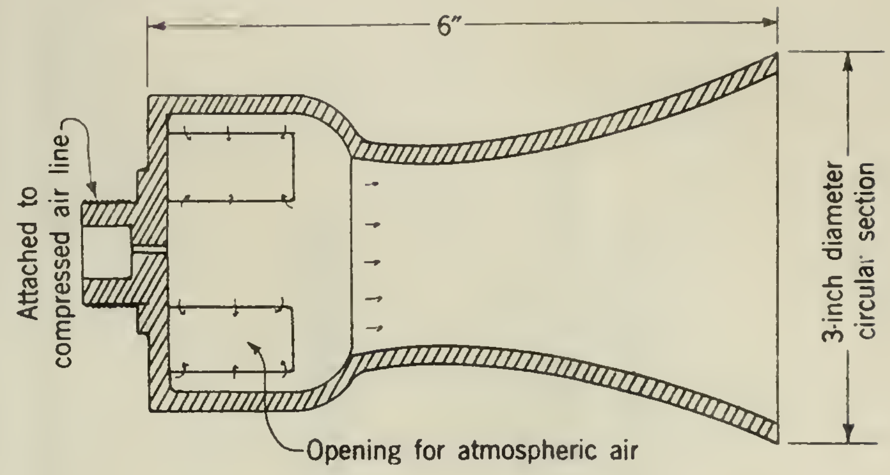

Figure 17.—Compressed-air blower nozzle

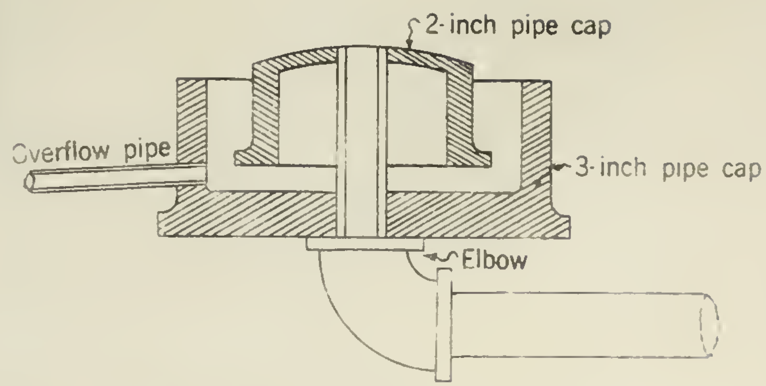

Figure 18.—Homemade drinking fountain

I. C. 6513 October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METHOD AND COST OF QUARRYING LIMESTONE AT THE QUARRY OF THE TRINITY PORTLAND CEMENT CO., FORT WORTH, TEX.

BY

J. William Ganser

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October, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF QUARRYING LIMESTONE AT THE QUARRY OF THE TRINITY PORTLAND CEMENT CO., FORT WORTH, TEXAS[1]

By J. William Ganser[2]

INTRODUCTION

This paper is one of a series being prepared by the United States Bureau of Mines describing mining and milling methods and costs at cement-plant quarries throughout the United States.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent local operating expenditures only and not total production costs. It is recognized that publication of total production costs may in some instances cause embarrassment to individual producers as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

The methods described herein are those used at the Fort Worth Plant of the Trinity Portland Cement Co.

ACKNOWLEDGMENT

In the preparation of this paper the author wishes to acknowledge the assistance of Fred Burkett, civil engineer of the company, who furnished the drawings, and of J. M. Simmons, chief clerk, who supplied the cost data. Bulletin 1931 of the University of Texas, "The Geology of Tarrant County," by A. M. Winton and W. S. Adkins, was consulted for information regarding the geology.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6513."

2 - One of the consulting engineers, U. S. Bureau of Mines, and chief chemist, Trinity Portland Cement Co.

HISTORY

The Trinity Portland Cement Co., of Dallas, operates three plants in Texas; one of which is in Dallas, one in Fort Worth, and the other in Houston.

In the spring of 1923 W. H. L. McCourtie, president of the Trinity Portland Cement Co., became interested in the possibility of building a cement plant in Fort Worth to supply the rapidly growing demand for cement in west Texas. After considerable prospecting south and north of Fort Worth, it was decided that the best lime rock and shale for cement purposes were north of Fort Worth.

A property consisting of 600 acres, about 5 miles north of Fort Worth was obtained and ground was broken in the fall of 1923 for the machine shop and stock room. In order to get a building on the ground quickly the old power house at the Dallas plant was dismantled and set up at Fort Worth. This old building had housed three 750-hp. Allis-Chalmers gas engines, which were being dismantled because of a gas shortage which resulted in the installation of a waste-heat power plant at Dallas. After the erection of this building at Fort Worth the work progressed slowly until the spring of 1924, when machinery began to arrive. Gradually during the following months the required buildings were erected and the machinery installed. Operation was started in the early part of May, 1925, and the first clinker was ground into cement on May 23.

In the layout of the plant the plan was to have all the buildings as close to one another as possible and to have everything compact so as to require the least amount of labor.

The plant is directly adjacent to the quarry which is situated on a gradual slope to a small creek which usually is dry during the summer. The only vegetation on this slope was sparse grass and cactus, and as there was little soil no stripping was necessary before opening the quarry.

GEOLOGY

The rock and shale are of the Washita division of the Comanchean series of the Cretaceous period. The shale or marl is called Kiamitia, and the rock is called Lower and Upper Duck Creek limestone. The highest points on the quarry have a thin layer of Fort Worth limestone.

These formations dip from the northwest to the southeast, at an inclination of about 50 feet to the mile.

The Kiamitia formation, as exposed under the Duck Creek limestone, is dark bluish, extensively laminated, and slightly calcareous. It is about 30 to 35 feet thick and where exposed weathers readily.

The following is an analysis of the Kiamitia shale found directly under the Duck Creek limestone:

	Per cent
Silica	52.80
Iron oxide	2.86
Aluminum oxide	14.24
Calcium carbonate..........	19.08
Magnesium carbonate	2.45
Loss on ignition	6.59

Only the upper one or two feet of the Kiamitia shale is quarried but it has been found through cores that the bottom portion, which is next to the Goodland limestone, is a dark blue clay marl with bands of sandy, calcareous material. In the middle section are thin limestone layers and several calcareous sandstone flag layers each 2 or 3 inches thick.

The Lower Duck Creek formation is a series of compact, soft limestone strata. It is about 35 feet thick, and is characterized by a remarkable sequence of zones containing ammonites. These limestone strata are separated by many seams of a soft limestone 2 and 3 inches thick which easily disintegrate.

The following is an analysis of the limestone taken from one of these seams:

	Per cent
Silica	21.80
Iron oxide	3.45
Aluminum oxide	13.15
Calcium carbonate	53.52
Magnesium carbonate	1.50
Loss on ignition	5.10

Analysis of the Lower Duck Creek limestone is as follows:

	Per cent
Silica	6.02
Iron oxide	1.29
Aluminum oxide	3.38
Calcium carbonate.........	84.96
Magnesium carbonate	1.75
Loss on ignition	1.50

The Upper Duck Creek limestone or limy marl, at the present quarry face, is about 10 feet thick. It is a light colored, yellowish marl containing a conglomeration of fragmental limestone and has the following composition:

	Per cent
Silica	17.70
Iron oxide	2.80
Aluminum oxide	9.20
Calcium carbonate	64.22
Magnesium carbonate	2.09
Loss on ignition	7.00

The quarry was first opened on the rim of an ampitheatre-like depression on the headwaters of a fork of Marine Creek which has cut back the Duck Creek and Kiamitia formations, thereby exposing the Goodland limestone in the creek bed.

METHODS OF PROSPECTING AND EXPLORATION

Previous to the purchase of the property considerable prospecting was done by persons expecting to interest someone in building a cement plant. What first probably attracted their attention was a 50-foot face of rock on Marine Creek about one-half mile north of the present quarry. Rock from this bluff was completely analyzed by us before we bought the property lower down the creek.

From the analyses we were convinced that Portland cement could be made from the rock in this locality. Two holes were drilled on the property, one 100 feet deep and the other 64 feet deep. The 100-foot hole showed 80 feet of limestone and marl and 20 feet of shale; while the 64-foot hole, which was drilled further down the slope showed 48 feet of rock and 16 feet of shale.

After the property was purchased we drilled six holes with a Cyclone drill to get more data as to the quality and thickness of the limestone. Samples from these holes showed that we had from 20 to 45 feet of rock over the area prospected. In each instance we drilled into the shale and all of the samples showed that it was quite uniform.

The following analyses were obtained on material from the deepest hole, each sample representing 5 feet of depth:

Analyses of material from deepest drill hole

Sample No.	Per Cent			
	Silica	Iron and aluminum oxide	Calcium carbonate	Magnesium carbonate
Rock:				
1	7.62	4.74	85.11	2.01
2	14.18	8.18	74.96	2.51
3	14.16	8.48	72.28	1.42
4	12.62	8.80	75.57	3.37
5	12.10	6.92	77.21	3.19
6	13.38	8.60	75.82	2.74
7	12.96	8.24	74.64	1.71
8	8.94	6.16	81.16	3.00
9	7.26	3.32	85.53	1.45
Shale:				
10	35.68	15.76	44.50	3.39
11	38.88	14.66	40.09	4.04

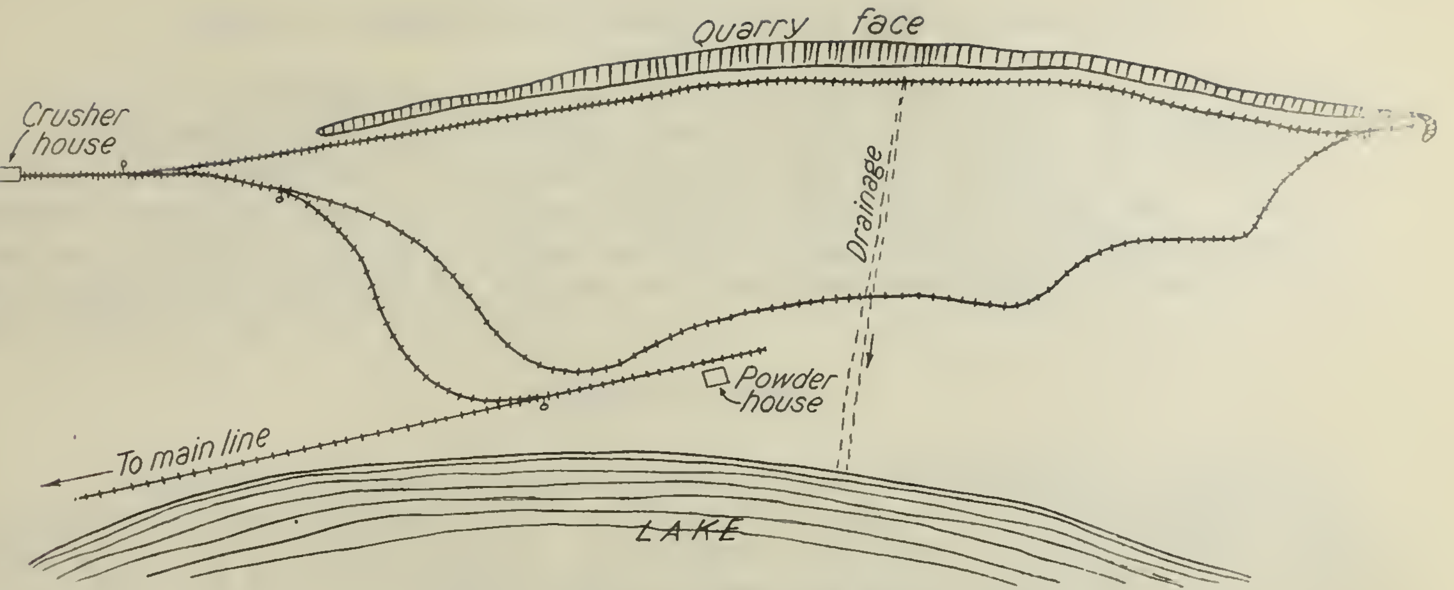

Figure 1.- Track layout and line of quarry face

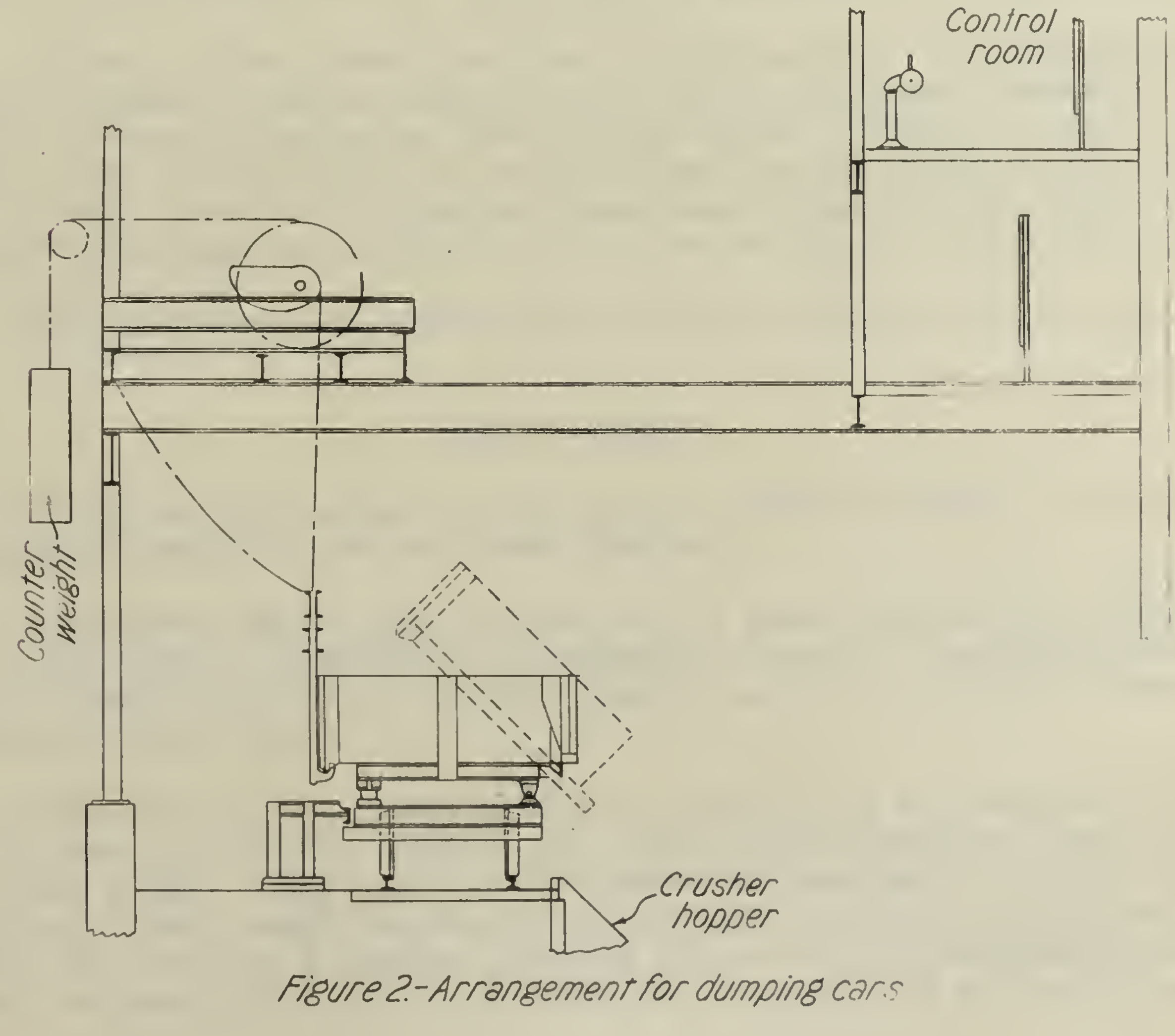

Figure 2.- Arrangement for dumping cars

METHOD OF SAMPLING AND ESTIMATION OF TONNAGE AND VALUES

In quarrying, little shale is mined with the rock and no shale pit is operated as is customary in other cement-plant quarries. The quarry face, which is 2,000 feet long, contains rock and marl low in lime at the end farthest from the plant and slightly high in lime at the near end. The low-lime and high-lime rock are stored separately and mixed by taking so many crane buckets of one kind to so many of the other and dumping them into the hopper which feeds the tube mill.

CHOICE OF METHOD

The quarry location had a gentle slope and therefore in opening the quarry it was only necessary to shoot the rock and head the shovel into a face about 10 to 15 feet high. At present the face is about 30 feet high at the ends and 50 feet at the center. The rock face is worked in a straight line. The overburden nowhere amounts to over 6 inches. Of the 600 acres of land owned by the company 400 to 450 acres are available for quarry purposes.

Drainage of the quarry has been an easy matter since the floor of the quarry is higher than the creek bed and it has only been necessary to dig shallow ditches to carry the water to the creek. While some water stands in shallow pools in the quarry, it only softens the quarry floor to a depth of about 1 foot. There is a stratum of hard shale about 1 foot under the limestone, which keeps the shovel and ties from sinking into the soft shale.

Figure 1 shows how the track is laid out and the line of the quarry face.

MINING METHODS

Since the overburden is only about 6 inches in depth it is not removed, but is mixed with the rock when blasted.

The quarry begins near the end of the plant and the face extends straight away from it; therefore, the haulage is short. The broken rock is loaded into cars by a No. 50-B Bucyrus electric shovel on crawler treads and having a 2-yard dipper.

When the plant is running at full capacity it is necessary that the quarry operate 10 hours per day, six days a week, producing from 1,200 to 1,500 tons a day. Only 6,000 to 8,000 tons of rock are shot down ahead of the shovel. The reason for this is to prevent the water-soaking of large quantities of rock in wet weather, because if the rock is thoroughly wet the marly part becomes sticky and it is difficult to handle in the preliminary crusher.

While the rock face is 50 feet high at its maximum it is never this high after being shot. An effort is made to shoot the rock so that it is not usually over 30 feet high in front of the shovel. The width of the shovel cut is about 40 feet.

DRILLING AND BLASTING

The rock is drilled in the usual way by putting down well-drill holes, for which a 4-inch Sanderson-Cyclone drill is used, electrically operated and on tractor wheels. This drill is operated by a 15-hp., 440-volt, 720-r.p.m., variable-speed motor.

The holes are drilled 14 to 16 feet from the face and down to the shale. Only one row of holes 16 to 18 feet apart is drilled at a time. These holes are shot one at a time, and only three or four are shot ahead of the shovel. This method of shooting was adopted to keep concussion to a minimum.

The drill bits are of tool steel, 4 inches wide with 45° slope on the drilling edge. Two men operate the drill 10 hours a day and drill about 120 feet. These men are paid by the hour, one receiving 40 and the other 30 cents. At this rate the labor cost is from $6\frac{1}{4}$ to 7 cents per foot drilled. As the holes vary from 20 feet to 60 feet in depth, the amount of dynamite put into a hole varies. The maximum amount put into a hole - a 60-foot hole - is one box or 50 pounds. All of the dynamite is put into the bottom of the hole, as it is the rock at the bottom that needs shattering. The top part - 1 foot to 10 feet - needs no shooting since it is a conglomerate of small rock and marl. When the bottom is shattered and pushed out by the shot, the top is easily broken up by the movement.

The dynamite used in these drill holes is 4 by 8 inch, 40 per cent gelatin. Cordeau-Bickford is used on all primary shots and is set off with a No. 6 detonator. The holes are all loaded in the usual manner, and tamped with dirt by a wooden pole on a rope.

Very little secondary shooting is necessary. The few rocks that are too large for the crusher are laid aside by the shovel and at some later time are shattered by "doby" shots. From one to four $1\frac{1}{4}$ by 8 inch sticks of 40 per cent gelatin dynamite are placed on the large rocks with a fuse about 30 inches long and covered with mud.

LOADING STONE

All the stone is loaded by a No. 50 Bucyrus electric shovel with caterpillar tractors. A 100-hp., 2,200-volt motor drives a generator which delivers 230-volt current to a 60-hp. motor on the hoist of the dipper, a 13-hp. motor which swings the shovel, and a 13-hp. motor on the thrust. A 5-hp. motor on the dipper trip, a 2-hp. motor on the air compressor and $7\frac{1}{2}$-hp. motor on the exciter are run by 440-volt current received through a transformer from the 2,200-volt leads. This system of motors is called the Ward-Leonard control. It is necessary to have only one man on the shovel and one man in the pit, who removes large rocks that might fall on the track and also signals with a flag to the tower man as to where to stop the electric car and when to move it after it is loaded.

When the quarry was first opened a No. 32 Marion steam caterpillar shovel, with a 1-yard dipper, was used. After several years this was too

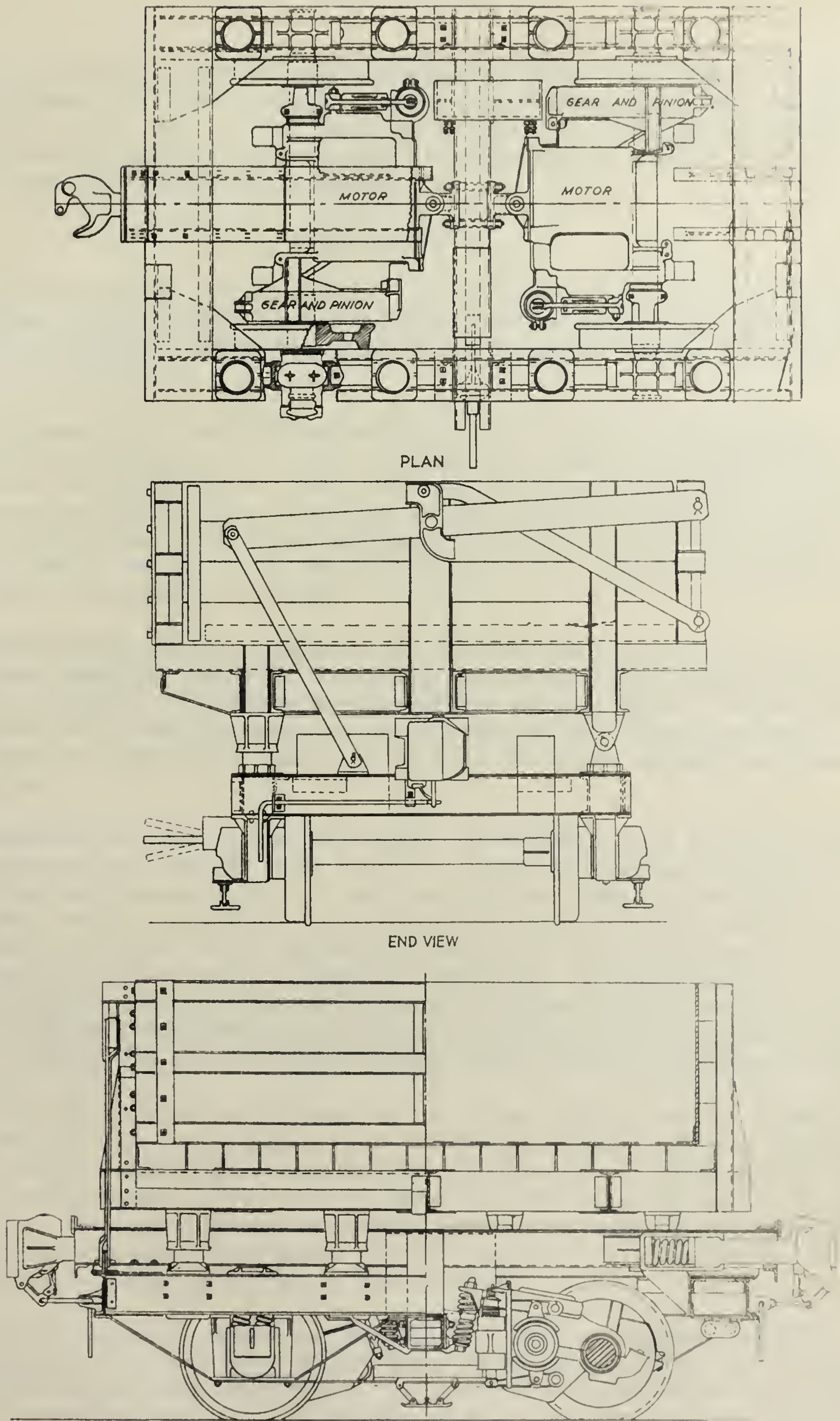

Figure 3.- Electric quarry car

small to supply the quantity of rock necessary, and the bank became too high to operate so small a shovel safely. Since the crushing plant was being run entirely by electric power it was then decided best to purchase an electric shovel, thereby eliminating the necessity of maintaining a water line in the quarry and also the purchasing of coal.

TRANSPORTATION

Transportation from the quarry to the plant is by Woodford (electric) haulage system. This system, is controlled by an operator in a control tower which, in this instance, is in the plant above where the cars are dumped into the receiving hopper.

The cars, of 10-cubic yard capacity, were built by the Western Wheeled Scraper Co. They are of composite wood and steel construction and are of one-way, side-dump type. Each car has two 30-hp. Allis-Chalmers motors. Each motor drives an axle through direct single-reduction gearing. Three cars are being operated and each is equivalent in power to a 35-ton locomotive, as they will easily switch a box car of gypsum to the place of unloading. Figure 3 shows the design of the cars.

The Woodford system employs a third rail using 250-volt direct current. In some installations this third rail is between the haulage rails, but at this property it is placed on the outside of the rails. It was found that the third rail, when between the two haulage rails, interfered when switching box cars; therefore the third rail on the whole system was placed on the outside.

Operation of the system is simple, as everything is in full view of the tower operator from the time the cars leave the crusher building until they are loaded and returned to it. Each controller provides for three running speeds, one off position for coasting, and three brake positions for various brake pressures. With the controller handle in the off position a car will coast until the brake current is turned on. The controls are set so that the operator need not attend to the cars except for a few feet at the loading and discharge ends where they require accurate spotting. The three-speed control gives the cars a range of speed from 12 to 16 miles per hour.

The track system is composed of one track going from the crusher house to a switch, and two branches from the switch joining to form a long loop. One side of this loop is parallel to the face and is kept near it to serve as a loading track. The other is away from the face and is called the passing track. The usual method is to send empty cars out over the passing track and around the loop, spotting a car at the shovel as soon as the car before it has been loaded and sent to the crusher.

The device which dumps the cars at the crusher is also operated by the haulage man. It consists of two hooks which are attached to a frame and yoke. Two chains from this yoke go to cam-quadrants on which the chains wind as the quadrants revolve. The weight of the yoke and hooks is balanced by counterweights fastened to wire ropes which are wound on sheaves on the same shaft as the quadrant. When this shaft is turned by a motor (through

gearing) the hooks are lifted, dumping the car sidewise, not only quickly but smoothly, and the car body is set back on the bolsters with very little jar. Figure 2 shows the design of the car-dumper.

CRUSHING PLANT

The cars are dumped into an 18 by 20 foot steel-lined hopper from which the rock is discharged by a pan-type feeder, driven by a 5-hp., 440-volt motor, to a No. 7 Mammoth Williams hammer mill, driven by a 250-hp., 2,200-volt, 750-r.p.m. motor. Feed to hammer mill consists of pieces ranging from a 3-foot cube to 1/8 inch in size.

The discharge from the Williams mill is elevated by a 42-inch pan elevator, driven by a 25-hp., 2,200-volt, 720-r.p.m. motor, and discharged onto a short belt 30 inches wide driven by a 5-hp., 440-volt, 720-r.p.m. motor, which carries the rock over a wall into the rock storage.

Following is a screen analysis of the discharge from the Williams mill:

Size of mesh passed,	Per cent
1½ Inch	89.5
1 "	84.5
¾ "	75.7
½ "	67.7
¼ "	59.0
1/8 "	30.2
20 Mesh	16.7
30 "	9.5
50 "	5.6
100 "	2.7

The rock after arriving in storage is either put into the section for low-lime or high-lime rock, or taken to the bins feeding the tube mills. The rock storage is 75 feet wide and 178 feet long and has a capacity of 10,000 tons. This rock is moved in storage by a Shepard 6-ton traveling crane equipped with a 3-yard clamshell bucket. This crane has a span of 75 feet and is equipped with a 10-hp., 1,200-r.p.m., 440-volt trolly motor, two 40-hp., 900-r.p.m., 440-volt hoist motors, and a 40-hp., 900-r.p.m., 440-volt, bridge motor. All of these motors have variable speeds.

As already stated, the quarry is arranged so that high-lime and low-lime rock mixtures may be made. These mixtures can be further varied by adding different proportions of the rock from the crane bin, all of which makes for uniformity and for less correction after the raw material has been ground into slurry.

The two tube mills for grinding the raw mixture are adjacent to the rock storage and are fed from a hopper loaded by the crane. They are 8 by 30 foot Traylor combination mills, each driven by a 700-hp., 2,200-volt, 180-r.p.m. motor. These mills each have three compartments, so that the

reduction is in three stages. The total weight of the grinding media per mill is 70 tons, of which 20 tons of 2 by 4 inch balls are in the first compartment, 20 tons of $1\frac{1}{2}$-inch balls are in the second compartment, and 30 tons of 1 by $1\frac{1}{4}$ inch slugs are in the third compartment. The balls consumed per ton of material ground amount to 0.304 pound. Tube-mill liners last from two to three years. The raw slurry from the mills contains 40 per cent of moisture, and 90 per cent of it will pass a 200-mesh screen. The two mills grind at the rate of 29.39 tons per hour, or 14.70 tons per mill per hour.

The slurry from the mills flows to a bucket-elevator pit from which it is elevated to silos, where it is agitated before being fed to the kilns and burned at 2,700°F. into Portland-cement clinker. Figure 4 shows the flow sheet from the quarry to the kilns.

WAGE SCALE

No regular foreman is employed in the quarry, the superintendent, or mill foreman, supervising when necessary. The quarry men receive wages as follows:

Position	No. of men	Wages per hour
Shovel operator	1	$0.65
Pitmen	1	.35
Driller	1	.40
Driller's helper	1	.30
Powderman	1	.50
Crusher-tender	1	.40
Crusher-tender helper	1	.30
Towerman	1	.60
	8	

SAFETY ORGANIZATION

The idea of safety is being brought to the men at this plant in an unusual but impressive manner. All those guilty of breaking the safety rules if found out appear at a court held each Monday morning and are tried for the offense. The following is a description of this court, by J. M. Simmons:

> A judge is elected to preside over the court, select the jury, appoint the prosecuting attorney, and perform such duties as befall a judge in regular court. A sheriff is elected, who, incidentally, is a man whose regular duties as an employee require that he go through the entire mill several times a day. It is the duty of the sheriff to see if he can detect any employee doing a careless act or anything that may endanger himself or his fellow workers. Should he discover anything of this nature he files charges with the court clerk, who issues a notice to the man to appear at the next meeting of the court to answer the charge turned in by the sheriff. The sheriff also notifies any witnesses that he is able to find. Any employee in the mill can file charges, through the sheriff and court clerk.

At five minutes to 10 o'clock each Monday morning, at our regular safety period, the plant whistle blows; indicating that the men in each department selected by the department foreman are to appear in the court room - our regular first-aid room. The foreman has no trouble whatever in getting the men to come to the court, for they would all come if allowed.

The meeting is called to order promptly at 10 o'clock, and the minutes of the previous meeting are read by the secretary. We then have a short discussion on ways and means of preventing accidents, asking for suggestions from the men. We get our best recommendations from the men who do the actual work where hazards exist. These recommendations are put in the minutes of the meeting and are read the following week, giving account of those carried out.

The meeting is then turned over to the judge (shipping foreman), who opens the court by giving a brief talk on the purpose of the court stating that it is not intended to create any personal feeling, but that it is all in a spirit of fellowship and good will.

The judge then has the first case on the court docket read by the clerk, asks for corrections, appoints a jury, swears in the witnesses, and turns the case over to the prosecuting attorney. The prosecuting attorney and the attorney for the defense (selected by the defendant) have 10 minutes each to question the witnesses of both sides and the defendant himself, and five minutes each to sum up and plead the case to the jury. When the defense rests the case, the jury retires to the jury room for not more than three minutes, after which time it returns a verdict of guilty or not guilty.

If the defendant is found guilty, the judge lectures the prisoner on safety work and particularly on safety work in line with the case just tried, which lecture incidentally is for the benefit of the entire court.

In the fall of 1929 practically the entire plant was given first-aid training by the U. S. Bureau of Mines. Since then six men have formed a first-aid team and have on several occasions entered first-aid contests in Fort Worth and Dallas.

Accident record of the plant since 1927

Year	Days lost per 1,000 man-hours	Man-hours worked
1927	0.528	261,542
1928	0.109	301,378
1929	0.403	431,670
1930	0.000	371,752

ADMINISTRATION ORGANIZATION

Figure 5 shows the organization chart for the entire plant.

Table 1. - Summary of costs

Trinity Portland Cement Co., No. 2. Period: Jan. 1, to Dec. 31, 1929.

Total material loaded during period: Overburden - - - included with stone
Stone - - - - - 294,883 tons (2000 lbs).

Operating costs per dry ton of stone mined

	Labor	Air drills	Power costs	Explo-sives	Other supplies	Total
Mining:						
Drilling	$0.0082	$0.0008	$0.0002	-	$0.0006	$0.0098
Blasting	.0058	-	-	$0.0250	-	.0308
Loading	.0203	-	.0015	-	.0075	.0293
Transportation	.0306	-	.0028	-	.0119	.0453
Crushing	.0139	-	.0087	-	.0121	.0347
Grinding	.0341	-	.1191	-	.0274	.1806
Elevating	.0029	-	.0005	-	.0048	.0082
Conveying	.0029	-	.0005	-	.0011	.0045
Miscellaneous plant	.0147	-	-	-	-	.0147
Total operating costs	.1334	.0008	.1333	.0250	.0654	.3579
Depreciation	.0300	-	-	-	-	.0300
Depletion	.0100	-	-	-	-	.0100
Taxes	.0100	-	-	-	-	.0100
Insurance	.0100	-	-	-	-	.0100
Miscellaneous overhead	.0200	-	-	-	-	.0200
Total costs	.2134	.0008	.1333	.0250	.0654	.4379

Table 2. - Summary of costs in units of labor, power, and supplies

Period: Jan. 1 to Dec. 31, 1929.

Material loaded during period: Overburden - - - - - - - included with stone.
Stone - - - - - - - - - - - - - - 294,883 tons.

	Mining	Crushing	Total
A. Labor (man-hours per ton):			
Drilling	0.0215		0.0215
Blasting	.0142		.0142
Loading	.0330		.0330
Haulage	.0534		.0534
Miscellaneous	.0111		.0111
Total labor	.1332		.1332
Average tons per man per shift			47.52
Labor, per cent of total cost			37.24
B. Power and supplies:			
Explosives (lbs. per ton)			.1639
Total power (kw.h. per ton)1/			14.0196
Shovels	.0015		.0015
Haulage 2/	.0028		.0028
Drills	.0008		.0008
Crushing		0.0087	.0087
Elevating		.0048	.0048
Conveying		.0011	.0011
Tube mills		13.9999	13.9999

1/ Average cost of power is $0.008 per kilowatt hour.
2/ Power includes that used on both locomotives and electric haulage system.

Table 3. - Detailed average shovel costs, direct operation

Period: Jan. 1 to Dec. 31, 1929.

Type of shovel, 50-B Bucyrus; size of dipper, 2-yard.

Stone loaded, tons 294,833.

	STONE COST	
	Total	Per ton
Engineers	$3,002.25	$0.0102
Pitman	1,394.35	.0047
Other operating labor ..		
Total operating labor	4,396.60	.0149
Fuel or power	454.12	.0015
Grease and lubricants	199.29	.0007
Coal - crane expense	735.50	.0025
Total supplies	1,388.91	.0047
Shovel-repair labor	1,587.40	.0054
Repair supplies	1,283.96	.0043
Total repairs	2,871.36	.0097
Total shovel operation	8,656.87	.0293

Table 4. - Detailed summary costs - direct operation

Period: Jan. 1 to Dec. 31, 1929.

	STONE COST	
	Total	Per ton
Total all shovels	$8,656.87	$0.0293
Drilling:		
Operating labor	2,357.20	.0080
Air, Gas, Power	69.90	.0002
Operating supplies	168.50	.0006
Repair labor	60.15	.0002
Repair supplies	229.00	.0008
Total drilling	3,723.23	.0098
Blasting:		
Labor	1,697.30	.0058
Explosives	7,372.71	.0250
Other supplies		
Total blasting	9,070.01	.0308
Haulage:		
Locomotives	3,533.40	.0119
Cars	1,531.31	.0051
Track maintenance	8,359.58	.0283
Haulage	13,424.29	.0453
Grand total	$34,036.00	$0.1152

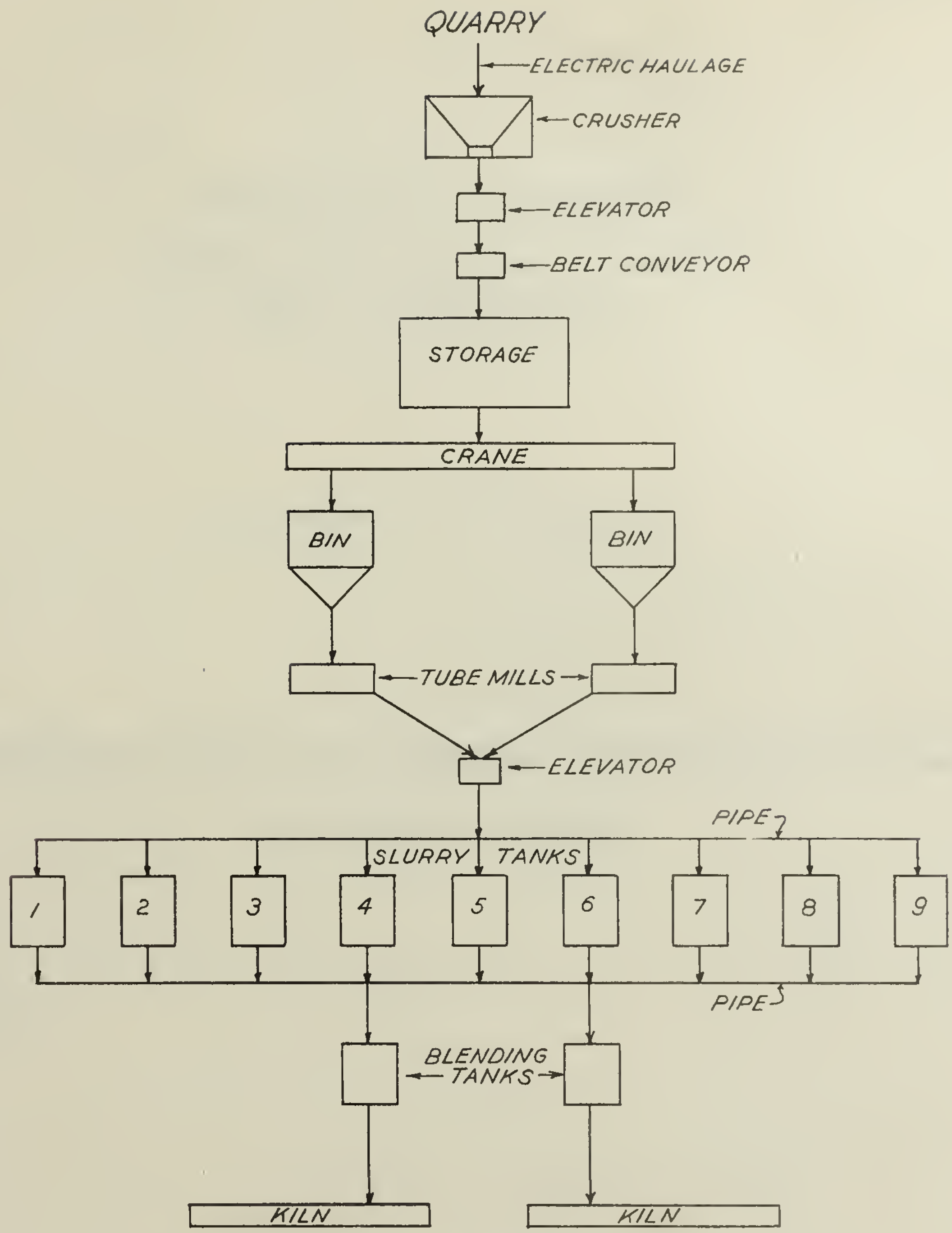

Figure 4.- Flow sheet

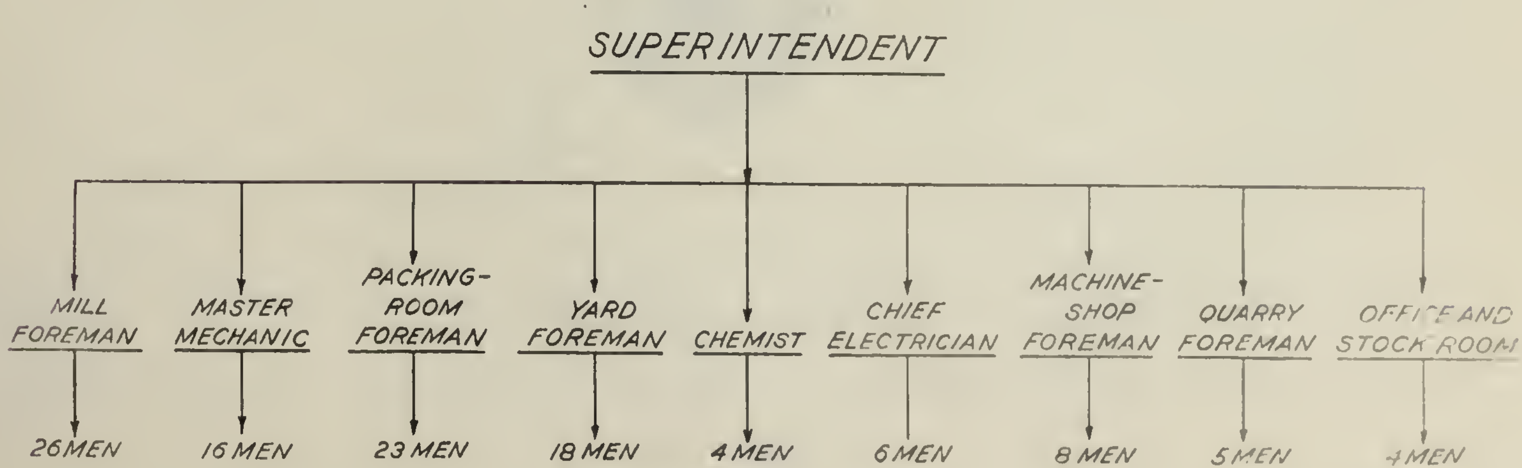

Figure 5.- Organization chart

I. C. 6514

September, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS OF THE MOLYBDENUM CORPORATION OF AMERICA AT QUESTA, N. MEX.

BY

J. B. Carman

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September, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS OF THE MOLYBDENUM CORPORATION OF AMERICA AT QUESTA, NEW MEXICO[1]

By J. B. Carman[2]

INTRODUCTION

This paper, describing the mining practices and methods at the mine of the Molybdenum Corporation of America, is one of a series of similar papers on mining methods in the various mines of the United States being prepared by the Bureau of Mines.

The mine of the Molybdenum Corporation of America is in the Red River mining district, Taos County, northern New Mexico (fig. 1). The mill and camp are in the valley of Red River, about 7 miles east of the town of Questa; the mine is 1¼ miles away, in Sulphur Gulch, a tributary of Red River. The mine is about 600 feet higher than the mill, at an altitude of about 8,700 feet. The climate is cold in winter, but does not interfere seriously with continuous operation. The mine normally produces about 40 tons of sulphide molybdenum ore per day, which is concentrated at the local mill and shipped to the plant of the Molybdenum Corporation at Washington, Pa.; there it is made into either ferromolybdenum or "molyte" (calcium molybdate), for use in alloys. The ore is mined from narrow, irregular veins by the cut-and-fill method. About 65 men are normally employed underground and on the surface at the mine.

The assistance of C. H. Johnson, assistant mining engineer of the United States Bureau of Mines station at Tucson, Ariz., in arranging the material in this paper for publication is acknowledged.

HISTORY OF PROPERTY

This property was first recognized as a potential producer of molybdenum in 1916 or 1917, when the metal was at a high premium. The story is told that the outcrop had been known of locally for a long time, but the molybdenite was mistaken for graphite, and the mineral had actually been mixed with grease and used as a lubricant by farmers nearby. Some claims were located during the war. The R. & S. Molybdenum Mines Co. was organized in November, 1918, by two men from Cripple Creek, Colo., and some ore produced which was milled at a gold mill on Red River about 5 miles above the mine. The Molybdenum Corporation of America was incorporated in 1920; in the same year it acquired this property, and operations were continued on a small scale until the market vanished in the general depression of 1921. However, a small amount of development work was continued, and in 1923 the construction of

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1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6514."

2 Manager, Molybdenum Corporation of America, Questa, New Mexico, and one of the Consulting engineers, U. S. Bureau of Mines.

the present mill and camp site was started. The mill began operation on November 1, 1923, since that time the mine has been worked continuously.

The production of the mine previous to the starting of the mill was about 250,000 pounds of molybdenum sulphide; including that amount, the production to January 1, 1931, has been approximately 5,000,000 pounds of molybdenum sulphide in the form of concentrates.

GEOLOGY

The rock in which the ore occurs is described as a soda-potash alaskite porphyry.[3] This is intrusive into a dark-gray granodiorite porphyry. The alaskite and the granodiorite are overlain by volcanic tuffs and flows, which are not present, however, in the immediate vicinity of the workings.

The mineralized veins are distributed very irregularly in east-west fracture zones which are sometimes several hundred feet wide. The veins consist of groups or sets of closely spaced, branching and interfingering fractures that are often separated from other vein systems in the same shear zone by many feet of barren rock. The fracture zones are much longer, as well as wider, than the mineralized veins found within them.

A system of north-south fractures offsets the veins in many places. These fault zones are entirely barren.

The only mineral of economic value is molybdenite (MoS_2), which occurs as a fissure filling in the east-west system of veins. The most important gangue mineral is quartz, with some pyrite, and minor amounts of biotite, fluorite, rhodochrosite, and calcite. The space between the small branching quartz-sulphide veins is occupied by altered, sericitized country rock; the vein filling actually comprises only a small proportion of the ground broken in mining.

No veins of economic value have been found in the granodiorite, and mineralization in this rock is limited to small stringers branching from veins on the contact between the alaskite and the granodiorite; such stringers never extend more than a few feet away from the contact.

The oxidation of the veins is very shallow, rarely extending more than a few feet below the surface, and generally some sulphide persists even at the surface. The broken ore in stopes does not oxidize rapidly enough on standing to injure it for milling treatment. The mine waters are alkaline, whereas if oxidation of the sulphides were taking place to any considerable extent, acid waters would be in evidence.

PHYSICAL CHARACTERISTICS OF ORE AND COUNTRY ROCK

The alaskite and the diorite, as well as the vein material, are very easy to drill and break. Development workings in the country rock do not require timbering. Drifts along the veins usually stand well and normally are not timbered before the start of stoping; about one-quarter of the drift footage is in heavily broken ground which requires support. Stulls are used for temporary support in some stopes, particularly the more flat-dipping

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3 Larsen, E. D., and Ross, C. S., The R. & S. Molypdenum Mine: Econ. Geol., vol. 15, November, 1920, pp. 567-573.

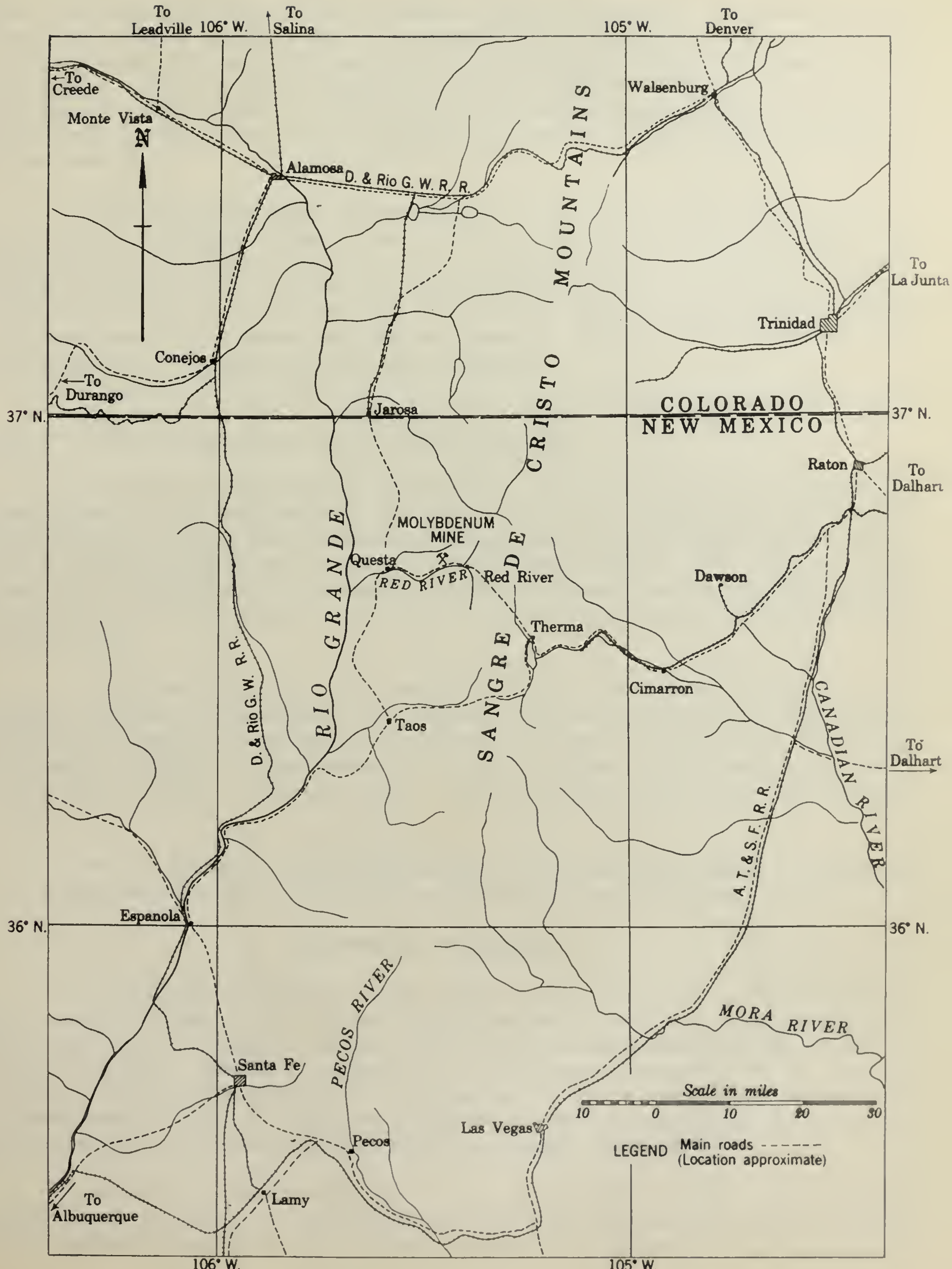

Figure 1.–Map showing location of mine of Molybdenum Corporation of America

ones; a few stopes have been worked out without support of any kind. The ground near the veins is not heavy in the sense that it "squeezes"; the chief difficulty in mining is caused by the often shattered and blocky nature of the ground where transverse slips have cut the main fissure. The granodiorite appears to be more fractured than the later intrusive. Sometimes the veins are largely gouge-filled, which increases the mining cost. The molybdenite itself is very friable, and large amounts of fines are produced in blasting unless it is done carefully.

Although the mechanical characteristics of the ore and country rocks offer no great obstacles to mining, the small size and irregular distribution of the ore bodies make them difficult to find and expensive to explore and develop. The largest continuous vein developed to date is about 1,400 feet long and 400 feet along the dip; the veins are commonly 200 to 500 feet long, with a length down the dip about one-third their horizontal length. The largest single stope extends about 240 feet along the strike and 170 feet along the dip; the average stope is probably not over half this size.

The dip of the veins ranges from 20 to 90° from the horizontal, and averages about 60°; it may change greatly or even reverse with depth.

The width of mineralization ranges from a fraction of an inch to 6 feet; sometimes the walls of the vein may be 10 feet apart, but in such cases the ore is confined to thin streaks close to the walls, with country rock between. For the whole mine the average width of material taken as ore is probably between 12 and 18 inches. Some relatively large stopes have been worked on veins averaging 4 feet in thickness. Ore of exceptional purity can be stoped profitably when as thin as 2 inches, and is commonly mined from veins not over 6 inches thick.

Transverse faulting has offset the ore so frequently that exploration is made very difficult, although the displacement is seldom more than 12 to 15 feet. The faulting and mineralization appear to have taken place together, although at more than one period. As the direction and extent of displacement are not at all uniform, it is impossible to predict in a given case where a lost vein will be found, or whether the ore ever extended any farther.

The ore shipped to the mill has a specific gravity of 2.56, which is equivalent to 12.5 cubic feet per ton in place, or about 19 cubic feet per ton, broken.

PROSPECTING AND EXPLORATION

The outcrops of the veins, being softer than the country rock, do not protrude above the surface. Where a vein outcrop is bare, however, as on a cliff or on the top of a sharp ridge, it is conspicuous because of the bright canary-yellow color of the molybdite, the usual oxidation product of molybdenite. The original prospecting of the property started by drifting on such outcrops, or crosscutting to them at lower elevations and then drifting along the vein. Following the exploration of one vein, crosscuts are usually run into the country rock in one or both directions to discover parallel veins.

Underground prospecting is done more or less blindly, although certain factors are recognized as guides. It is believed that economically important ore bodies will be found only near the margins of the alaskite porphyry. It has been noted that the ore is

most abundant where mineralized fractures are intersected by fissures of nearly the same strike but usually different dip. Rolls in the fault planes are often found to contain good ore. The best ore is usually found near the edge of an ore shoot, where the lens of sulphide is thick enough to be mined cheaply, yet not thick enough to include streaks of waste. A "fresh" appearance of the molybdenite is encouraging, and a stringer of such material is followed for a considerable distance, especially after commercial ore has been found in one or more places along it.

Stoping work is forced to carry part of the burden of prospecting, and at times is prosecuted in veins which are too narrow to pay, because of the desirability of following all mineralization to its end. Raising in ore is usually not done ahead of stoping, although in new country, a raise is sometimes run to determine the vertical extent of minable ore.

Because of the irregular dip of the veins, as well as the patchy distribution of the ore shoots, the downward projection of ore from level to level is not always successful.

No diamond drilling or long hole drilling is done because these methods of prospecting are not suitable to the ore occurrence; the ore shoots are so scattered that negative results from drilling would be untrustworthy and any ore found would have to be checked by later development. Moreover, the difference in cost between the footage of drilling that could be carried economically by the operation, and any similar amount of drifting, is not sufficiently large to make diamond drilling attractive. In the stopes, however, if there is reason to suspect that a branch vein exists close to one or another of the apparent walls, stoper holes are drilled into the walls and the cuttings inspected for ore.

Geophysical prospecting was done on the property in the fall of 1927.[4] Only a small portion of the indications have been tested. Good ore has been found in some places where it was strongly indicated by the survey. In one case good ore was found, not exactly at the indicated spot, but within 40 or 50 feet of it. In a few places ore has not yet been found, but in no such case has the possibility of ore been disproved. It is felt that the survey was decidedly a worthwhile investment.

SAMPLING AND ESTIMATION

The grade of ore is estimated with fair accuracy by inspection. Sampling of ore in place is done only in a few cases to check visual inspection as a guide to stoping, or rarely where the material is low-grade and an assay is desired to distinguish ore from waste. No samples of broken ore are taken at the mine. The ore from each stope, which is kept separate for purposes of contract payment, is grab-sampled as each truck load is weighed at the mill. This grab sampling is checked by an automatic sampler in the mill and is found to be sufficiently consistent for the purpose of mine operation.

Estimates of ore in a given vein are based on the appearance of the vein in drifts or stopes, and on past experience with similar-appearing veins in nearby parts of the mine. Such estimates of "probable ore" are not often correct as applied to short distances along a vein, but when applied to lengths of two or three hundred feet they have proved to be very

4 Sundberg, Karl, and Nordstrom, Allan, Electrical Prospecting for Molybdenite at Questa, New Mexico: Geophysical Prospecting, 1929, Am. Inst. Min. and Met. Eng., pp. 125-137.

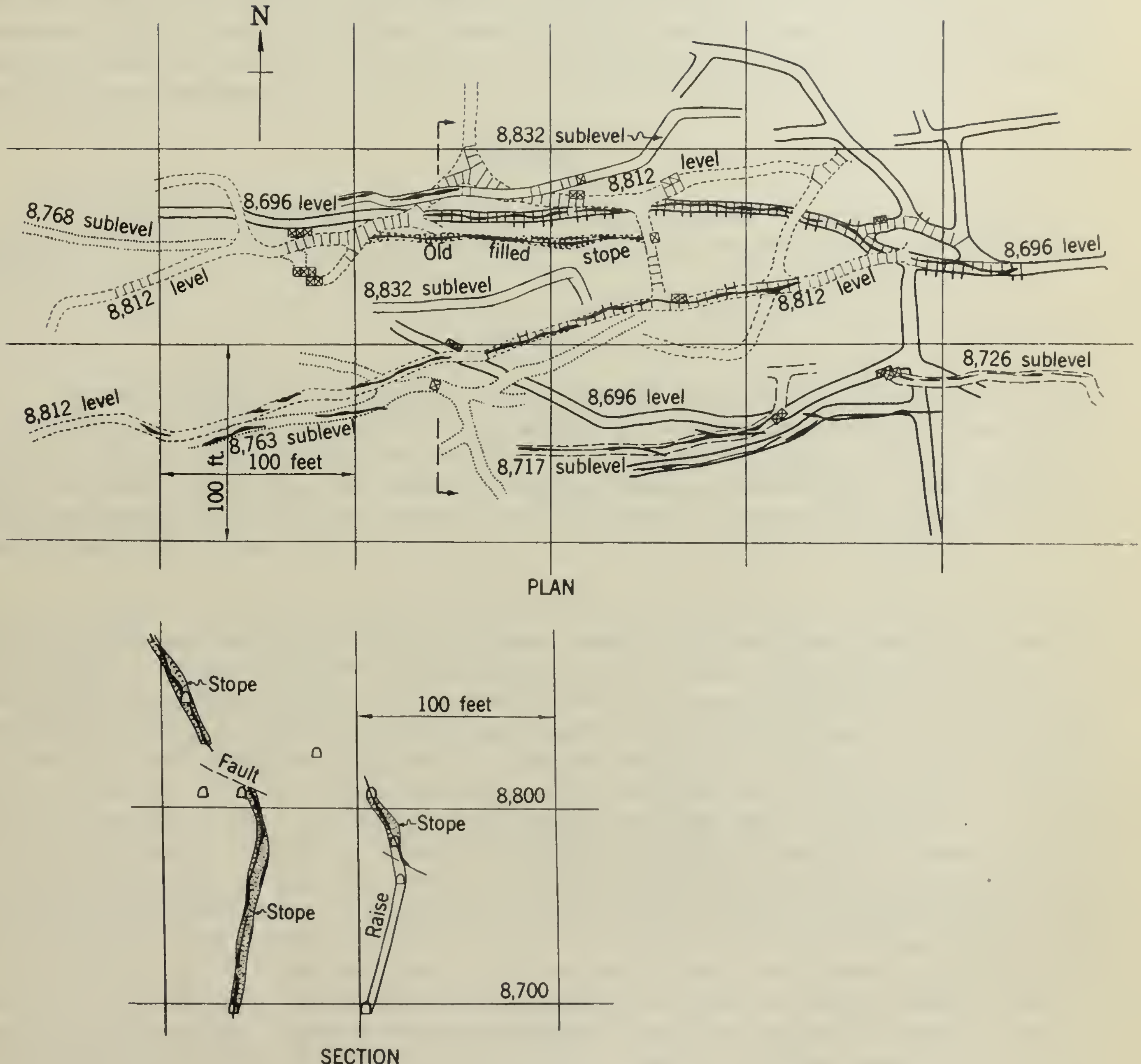

Figure 2.—Plan and section of portion of mine, showing method of development

close to actual extraction. The assumption is made, and with apparent justification, that veins of similar appearance, as regards thickness and character of mineralization, and located in geologically similar position, will produce amounts of ore in proportion to their areas - that is, length times width in the plane of the vein.

It is necessary to keep prospecting work spread out, in order to develop new areas and keep new stopes starting. Furthermore, stoping must be similarly spread out, to avoid the irregularity of production which would result if production were obtained by the concentrated, rapid mining of only a few stopes at a time. Both of these requirements are reflected closely in the cost of mining, but have been justified by the regular production which has been maintained.

DEVELOPMENT

The mine is developed by a number of adits and tunnels at several different elevations (fig. 2). The vertical interval between levels on the same vein ranges from 40 to 100 feet.

Drifting

One untimbered 5 by 7 foot adit several hundred feet long is larger than the usual drift or crosscut, having been driven with a view to its future use as a main haulage level. The ordinary drifts or crosscuts are 4½ by 6½ feet in section, and are driven untimbered, except in some places along the veins or in badly fissured country rock. The amount of drift and crosscut which must be timbered when driven because of heavy ground conditions is about 10 per cent of the total .

Machine drilling is done with a light drifter-type machine. Mounted jackhammers have been used, but were not adapted to the work; repair costs were higher, and poorer rounds were drilled. Seven-eighth-inch, hollow-hexagon, carbon steel is used with the drifters. The bit is the ordinary cross bit with a double 5° and 14° taper. Starter bits are 2¼ inches in diameter with gage changes of ⅛ inch and length changes of 18 inches. A few 135-pound drifters are used which were salvaged from another property. These take 1-inch, hollow-hexagon steel; bits for this steel are the same as described above. Line oilers are used with all machine drills. The machine-drill round ordinarily used in drifting is a 7 to 8 hole toe-cut round, drilled from a 3-inch horizontal bar over the muck pile. When properly drilled it breaks an average of 4 feet. The holes are loaded with 1 by 8 inch, 40-per cent strength gelatin dynamite; 4 or 5 sticks are used in the cut holes, 4 in the lifters, and 3 in the breast or back holes, or a total of 26 or 27 sticks per round. The explosive is detonated with No. 6 or 7 caps and fuse. A 50-per cent strength bulk powder in 1 by 8 inch cartridges, running about 50 per cent more sticks per box than the regular gelatin powder, was tried; it was found to produce marked economies in operation, because the miners used no more sticks than formerly and often actually less. However, it deteriorated very rapidly because of the lack of perfectly dry storage places at all of the scattered workings, and its use had to be discontinued. No stemming is used by the contractors, either on development or stope work.

Shoveling is all done by hand into 12 or 14 cubic foot cars; the broken material is trammed by hand to the surface or an ore pass.

Drift sets are stood on $5\frac{1}{2}$-foot centers. Posts are of 7-inch and caps of 8-inch round timber; collar braces are of 3-inch round and lagging of 3-inch split material. The poosts are 6 feet, 2 inches long, and are stood 4 feet apart at the top and 4 feet 4 inches at the bottom. The timber is mostly native Douglas fir and a little white spruce. It is being cut at present from burned-over areas, is well seasoned and gives satisfactory service. No preservative is used.

A drift crew consists of a miner and a mucker, working on the same shift. Development work may be done on either day or night shift, or both.

All development work and stoping are done on contract. Conditions vary greatly, and most of the details of drifting practice are not standardized, but are varied according to the ideas of the contractors.

A considerable amount of drifting in soft ground is done with hand drilling. In this case the work is organized differently. Each man works alone, breaking, mucking, and tramming. Drill steel is $\frac{7}{8}$-inch octagon, sharpened to the regular chisel bit. Starter bits are $1\frac{1}{4}$ inches wide, and four gage changes are provided of 1/16-inch each. The longest steel is about 4 feet long. Four-pound single-jack hammers are used. Hand-drilled rounds average less than $2\frac{1}{2}$ feet in depth, and have no standard form; the miner places his holes and points them to take every possible advantage of slips and joints in the ground. The average hand-run drift is less regular in shape than the machine drift, and is smaller, being about 4 by 6 feet in section. Occasionally for prospect purposes a hand drift will be run barely large enough to work a wheelbarrow, but usually regular track is laid and the broken rock trammed in cars. Seven-eights by eight inch powder is used for blasting. The economy in the use of powder resulting from the better placing of holes is shown clearly in Tables 1 and 2 at the end of this paper.

The rate of advance by hand drilling is slower than with machine drilling. However, two expert men, working two separate faces on the same contract, made an advance of 124 feet in 52 working days, or an average of 1.2 feet per man-shift. The drifts in this case were 4 by 6 feet in section. The men shoveled and trammed their own muck a distance of 600 feet.

Raising

About a fifth of the total development footage consists of raising. Raises are of two general types - cribbed or stulled, and may have one or two compartments. They are almost always inclined.

Hand drilling in raises is similar to that described under drifting.

Machine drilling in raises is done usually with 89-pound hand-rotated stopers, using 1-inch solid cruciform steel. This steel is sharpened with the same bits, gage, and length changes as the drifter steel. A few 110-pound wet self-rotated drills are used for raises of more than usual height, such as, for example, a 150-foot, three-compartment raise which was driven for a manway, air raise, and ore pass. These machines use 1-inch hollow hexagon steel. Center V-cuts with eight holes are used in two-compartment raises, and end cuts in single compartment raises. Blasting practice is the same as for drifts.

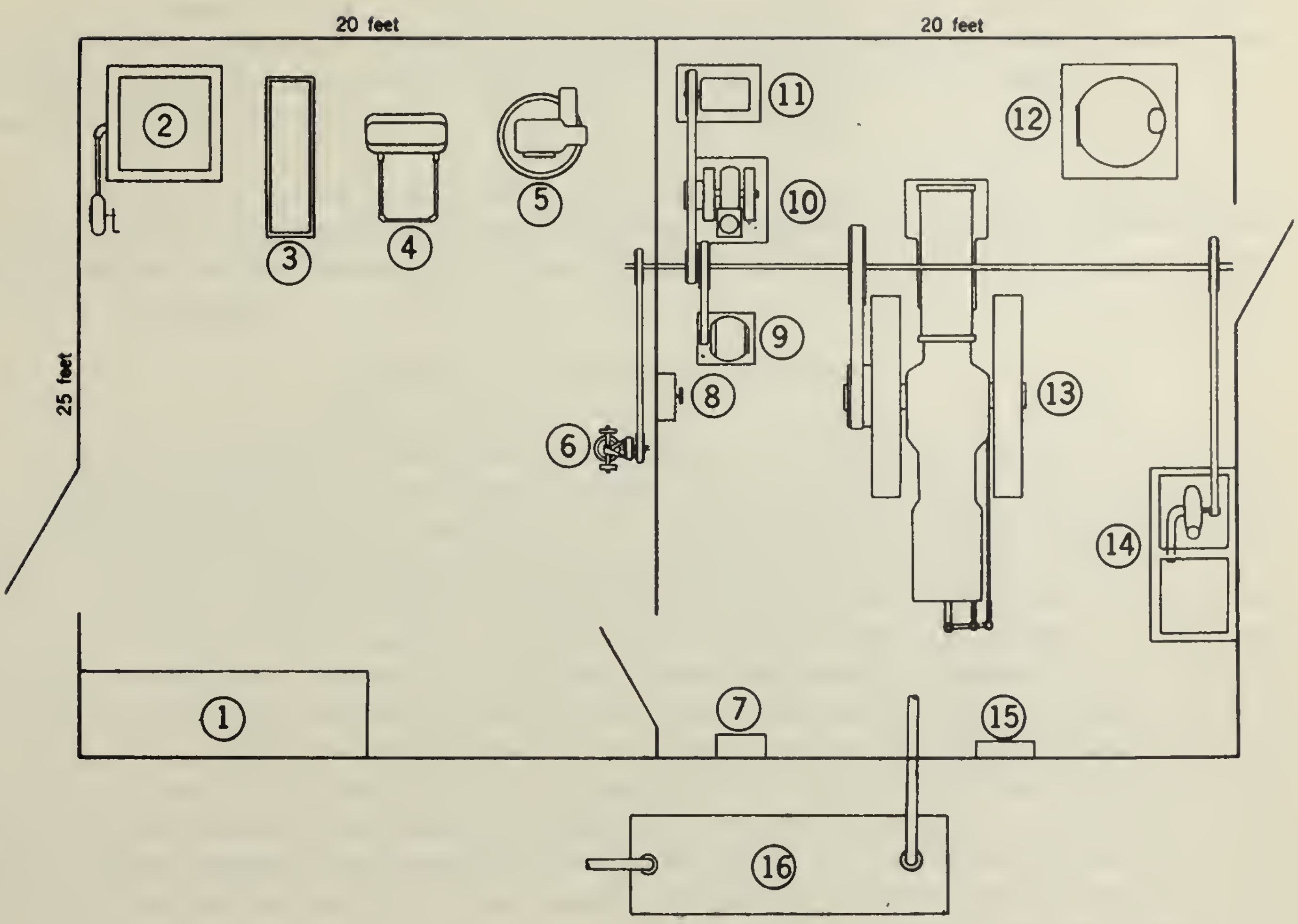

Figure 3.—Mine blacksmith shop and compressor house

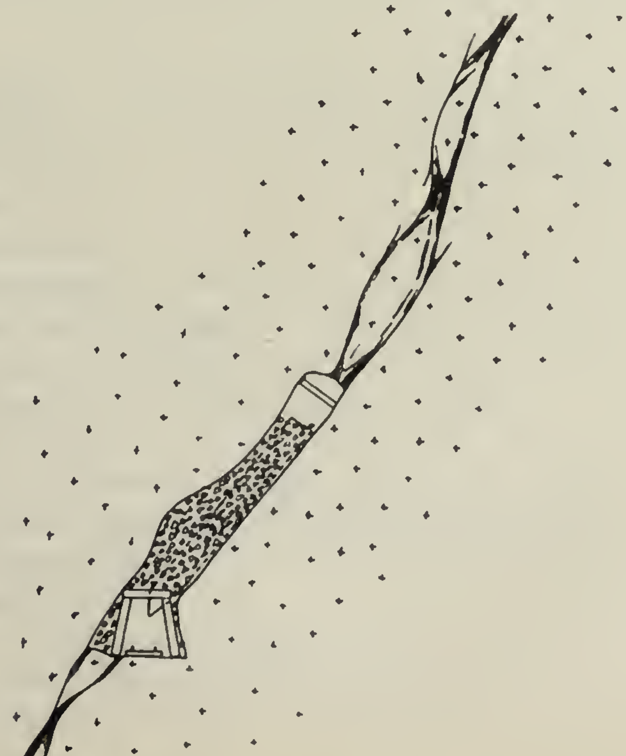

Figure 4.—Section through typical stope

A two-compartment stulled raise is ordinarily timbered with a single row of 7-inch round stulls placed from foot to hanging wall at 5-foot centers along the center line of the raise. The stulls are lagged on the chute side with half-round 3-inch lagging. Each compartment is 3 by 3 feet square.

Cribbing for raises is made of about 6-inch round timber, dapped at the ends and center to give 2 to 4 inch spacing between cribbing. The center cribbing in a 2-compartment raise is lined on the chute side with plank or split lagging to prevent rock from working through into the manway.

COMPRESSED AIR AND STEEL SHARPENING

Air for drilling is furnished by a compressor driven by a 17 by 19 257 r.p.m. 110-hp. full Diesel engine. The compressor is 2-stage, having a $16\frac{1}{2}$ by $13\frac{1}{4}$ inch differential piston connected tandem with the engine piston. The compressor has a sea-level rating of 600 cubic feet of free air per minute, but is only 75-per cent efficient at this elevation; it delivers 450 cubic feet of free air per minute at 90 pounds per square inch of pressure. The compressor serves, at a maximum, five drifter machines and five stopers. The stope drilling is intermittent, so that all machines are seldom working together. The pressure at the drills is about 80 pounds per square inch. The compressor discharges into a 4 by 10 foot receiver from which the air is taken to the portals of the main adits through a 4-inch line 2,200 feet long. The underground air lines are of 2-inch pipe. Two semi-Diesel 9 by 14 by 9 inch compressors, which formerly supplied the mine air, are kept in working order as stand-bys. These together are capable of delivering about 350 cubic feet per minute. The mine air line drains readily and the air passing through it becomes colder than the underground mine temperature, so that no trouble is experienced with freezing of air lines. Two additional receivers of about the same size as that at the main compressor are connected into the air line, one at the reserve compressors, and one in an intermediate position.

Figure 3 and the following list show the equipment and layout of the main compressor and blacksmith shop.

1. Work bench,
2. Coal forge,
3. Quenching tank,
4. Oil forge,
5. Drill sharpener,
6. Grinding wheels,
7. Recording pressure gage,
8. Rheostat and voltmeter,
9. D. C. generator, 1.5 kw., 115 v., 13 amp., 1,900 r.p.m.,
10. Water-cooled gasoline engine, 6 hp., 450 r.p.m.,
11. Starting-air compressor, $3\frac{3}{4}$ by $3\frac{5}{8}$, 475 r.p.m., 250 pounds pressure,
12. Coal heating stove,
13. Oil-engine-driven compressor,
14. Cooling water sump and return pump,
15. Lubricating oil filter,
16. Receiver, 4 by 10 feet.

The Diesel is ordinarily used to pump up the 375-pound starting air tank. However, if this fails to start the engine, the Fairbanks-Morse gasoline engine is used to refill the tank. This is belt-connected to a small 2-stage compressor. The Diesel and gas-engine are connected to the overhead shaft in such a way that they can be used independently as desired. Water from an outside tank above the compressor house runs by gravity through the compressor cylinder jackets and intercooler, thence through the Diesel engine water jacket, and into a small sump in the floor of the compressor room. From that point it may be either run to waste or returned to the supply tank by a 2-inch centrifugal pump, belt driven from the overhead shaft.

Machine steel is sharpened in a compressed-air sharpener; an oil-fired forge is used for heating and tempering. A small coal forge is used for sharpening hand steel; this is connected for blowing by hand or with compressed air. Water for the compressor and shop was developed by driving two short tunnels and building a retaining dam in the bottom of an apparently dry wash above the blacksmith shop. Water for drilling is collected in small sumps in the mine, and is supplied to the machines by portable water tanks.

One man on each shift operates the compressor, sharpens steel repairs drills and hose, and does general blacksmith work.

STOPING

About nine-tenths of the ore mined is extracted by horizontal cut-and-fill stoping and one-tenth by open stoping. The proportion varies slightly from time to time. Usually the ground requires the support given by filling. Other factors which may favor the cut-and-fill method are (1) the frequent necessity of breaking much more than the width of the ore to provide working room, in which case the resulting waste is most cheaply disposed of as stope fill and (2) the occasional very long tramming distances from stope to surface ore bin, which causes the contractors to do practically all of their sorting in the stope, the rejected waste then being used as fill. Open stoping is used where conditions permit, chiefly because it insures complete recovery of ore; particularly where the walls are of relatively hard, firm rock, there is considerable shattering of the ore in blasting, with increased danger of loss of fine material in the fill. The contract price of stoping is the same for filled and open stopes.

In both methods the ground is broken by overhand mining. In only one or two places the walls and ore have proved so soft and weak that overhand stoping was impossible. In these cases the ore was broken by breast stoping, the back supported by horizontal stulls and lagging, and the stoped ground filled closely.

Cut-and-fill stoping

A cut-and-fill stope is started by taking a cut from the back of the drift and placing drift sets and lagging, and then building chutes. In this work an effort is usually made to avoid loading much ore or waste by hand, to avoid blasting directly down on unprotected timbering, and to place the waste fill as soon as possible in order to keep within working distance of the back. In some places, a 5 or 6 foot cut is taken from the back of the drift for a distance of perhaps two sets. This is loaded by hand into cars, and sets stood and lagged over. The next round is drilled and blasted so as to throw most of the ore

over onto this timbering, where it can be sorted, and the ore dropped through the lagging into cars. Another set is then stood and the process repeated. In this way little hand loading is necessary and a fair covering of waste is left to protect the timbering when taking the next cut from the back. Another method of procedure is to break down a short 2 or 3 foot round from the entire length of the drift back to be included in the stope, then, working from on top of this broken material, drill a second round. Before shooting this, however, the first round is mucked out and timber stood. Then the second row is shot a few holes at a time and the broken ore sorted and dropped into cars below. The timbering under a stope is usually standard drift timber with single lagging overhead. In narrow stopes where the walls are good, stulls are occasionally used, or, perhaps, half sets, in which the hanging-wall end of the cap is supported on a post and the other end by a niche on the footwall.

Chutes are next built at 50-foot intervals, with manways at every other one. If a stope is less than 50 feet long, only one chute may be required. The loading chutes are of 2-inch plank, with double-plank control gates held in cleats. The ore chutes and manways are carried up by cribbing or by setting stulls from wall to wall and lacing them on the outside with split lagging. In 2-compartment raises the chute compartment is lined with plank on the manway side to prevent fine ore from running between the cribbing or lagging. The single compartment chutes are usually about 3 feet square inside; the cribbing may be made of timber as small as 5 inches in diameter. The tonnage handled through stope chutes is very small and the ore is not hard or abrasive. Consequently the lightest construction that will safely hold the weight of the fill is permissible and economical. As all ore is worked over by hand in the stopes, no grizzlies are used over the tops of the raises.

Although ventilation or prospect raises are sometimes driven from level to level in ore and later made use of in stoping or may be driven for these purposes after stoping has started, no raises are driven from level to level for purely stoping purposes, such as travel ways or for lowering supplies into stopes. This has been done in the past, but it was found very difficult or impossible to break the ore cleanly in a raise face, and the cost of the raises seemed excessive for the advantage gained.

Drilling and blasting account for only a small part of the labor in stopes at this mine. The ground is broken by stoper holes drilled almost straight into the back. Only a short length of back is drilled at a time, and the depth is seldom over 3 feet; where hand drilling is done, even less depths are broken. An effort is made to employ hand drilling in the stopes in soft ground, as recovery of the ore is cleaner and more complete. Hand drillers give much more regard to the condition of the ore or walls or the presence of cracks or slips; moreover, much less powder is used. Machine drilling is done with 89-pound hand-rotated stopers, using the same steel as described under "Development." Blasting is done with 40-per cent strength gelatin dynamite and No. 6 caps and fuse. Shooting is usually done at the end of the shift, but if ventilation is good it may be done at any time.

Various methods of breaking the ore are used, depending on the ore occurrence and the ideas of the foreman and miners. Figure 4 gives a cross section of a typical stope, showing the extreme variation in width of ore and occurrence of included waste. Where the ore is of sufficient width to give working room, and no breaking of the walls is necessary to fill the stope (that is, sufficient fill would be obtained by sorting), then the ore is broken down and the wall left untouched. If, as more commonly happens, the ore is not wide enough to provide working room, the practice is to break the footwall away from the ore,

sort out of this waste such ore as has come down with it, then take down the ore exposed on the hanging wall. This can be done by picking or very light shooting. Shooting holes in the ore streak is objectionable because it tends to pulverize and scatter the ore. If the ore is too loose to stick to the hanging wall it must be brought down with the waste and sorted. If the ore is in streaks along both walls, as often happens, the waste between can sometimes be mined first, and the ore afterward. The ore occurrence is so irregular that stoping practice may have to change almost every round.

Loose parts of the back are temporarily supported by stulls from foot to hanging wall. The contractors are required to cut, handle, and place this timber without extra compensation.

Plank flooring is sometimes laid on the fill before shooting down ore, to prevent the loss of ore in the fill. (It is particularly advisable if the waste fill is composed of coarse material.) This is not the usual practice, however, and the fill is simply leveled, and perhaps covered with a surface of fine waste before blasting. Dry-walls are sometimes built to provide level floors when part of the stope is higher than the rest; this avoids scattering of ore down the slope.

The ore is shoveled by hand into the chutes. If the distance to a chute is greater than usual, wheelbarrows may be used, but this is not necessary for the ordinary 50-foot spacing of chutes. Double handling is usually necessary in any case because of the sorting. In flat stopes some shoveling or scraping may be necessary to move the ore down the chutes, as it will not run on less than a 40° slope.

Stope backs should be kept horizontal. The tendency of the stope contractors is to push the wider and richer faces ahead and slight the narrow parts of the vein. An irregular back leads to operating difficulties and if allowed to become too much so there is the possibility of overlooking and losing good ore. If the ore pinches persistently in one place, that ground is sometimes left in place as a pillar. No known minable ore is left in place, and no pillars are left under or over levels.

Some sorting is almost always done in stopes. It may consist of merely picking out coarse waste, if the ore can be broken clean. Otherwise, the broken material may be shoveled onto an inclined screen of 2 or 1 inch wire mesh, or rarely ½ inch. The size screen used depends on the nature of the material, the idea being to pass most of the ore and exclude as much waste as possible. If much ore occurs in chunks which would not pass through the finer mesh, the 2-inch screen would be used. Even then the oversize would perhaps have to be picked over by hand for large chunks of ore.

If the ground is soft and breaks finely and the ore is scattered so that little volume can be eliminated by sorting or screening, then stope sorting may be omitted, and this work be done at the surface. Rarely the ore can be broken so cleanly and be of such grade that it can be shipped to the mill without any sorting or screening either in the stopes or at the ore bins.

The amount of waste sorted in stopes varies greatly, according to the mining scheme and the ore. Most of the ore receives some additional sorting at the surface, as will be described later; it is estimated that 60 per cent of the material broken as ore is rejected by sorting or screening and left in the stopes and that 10 per cent of the remainder is rejected by sorting at the surface.

Two to four men work in a stope, one or two of whom are contractors. Drilling, sorting, timbering, and mucking may be done indiscriminately by these men as the need arises. They also deliver the ore to the loading bins and sort it there.

Figure 5 shows the general design and construction of a typical sorting bin such as is built at the portal of each important producing adit. The track from the mine runs over the top of the bin, which in the case illustrated has four compartments. With the screen arrangement shown, the bin could care for the output of two contracts. Ore may be dumped directly into one of the unscreened compartments if it does not require sorting. Otherwise it is dumped onto a screen. The oversize slides onto the sorting platform where pieces of ore are picked out by hand and thrown into the adjacent bin. The waste is then shoveled into a car on the waste track and trammed to the dump. The sorting is done by the contractors as part of their stoping contract.

Open Stopes

Open stopes are usually small and are usually in flat dipping veins. The method is often applied to ore which is known to be of slight extent or to ore whose extent is doubtful and where it is desired to explore in this way before installing chutes and regular drift timbering. In the latter case the ore and waste are shot down onto the drift floor and trammed to surface for sorting. When such a small stope opens up it may be timbered or converted to a fill stope. When the ore in a regular filled stope splits, open stoping may be used to follow it for a short distance to avoid the necessity for building chutes and manways up through the fill.

The chief difference between open stoping with stulls on a larger scale and fill stoping is that more stulls may be used and that most of the material broken is removed; some sorting, however, is almost always done in the stopes and the waste gobbed in the bottom of the stope, so that the hanging wall is actually supported by this fill to a coniderable extent. The method of breaking ground is the same as in a filled stope.

TRANSPORTATION

The ore is trammed to the surface and dumped into the bins by the stope crews. The cars on most levels are 12 or 14 cubic foot, box-body, end-dump type, weighing about 700 pounds and holding about 1,500 pounds of ore. The newer cars are provided with roller-bearing cast-iron wheels. They have a step hook to permit swinging and dumping and a lever to release the end door. The 12.3-cubic foot cars are 24 inches by 48 inches inside, 18⅝ inches deep, and are 34½ inches high above the rail. On two levels larger cars are used. The long crosscut mentioned under "Development" is equipped with 16-cubic foot, box-body, end-dump cars, fitted with roller-bearing wheels.

Track is of 8-pound steel rail laid to 18-inch gage on 4 to 6 inch ties of round timber slabbed flat on top and bottom. The track in the long crosscut is of 12-pound rail, and this size will be used to replace the present track in one other level where the 8-pound track is proving too light for the 16 cubic foot cars used there.

The hauls from stope to ore bin range from 200 to 1,000 feet, averaging about 800 feet.

Ore from some of the higher and more remote tunnels, comprising about 15 per cent of the total mine output, is transported to the truck-loading bins by burros. This work is contracted to one man who operates a string of one or two dozen burros. The miners at these workings sack the ore in 70-pound sacks. Three sacks are usually loaded on a burro on regular pack saddles. The distances this ore is packed are not great, seldom over a few hundred feet in a straight line, but the drop is often two or three hundred feet. The burro man is paid a flat rate of 5 or 6 cents per sack, regardless of the amounts or distances packed, except in rare special cases.

Trucking from the mine to the mill is contracted to one man who operates two 1-ton, dump-body, automobile trucks and has a third truck in reserve. The average load of ore is very close to 5,000 pounds. In spite of their overloading, the trucks have stood up well and during their two years of operation the repair cost has not been excessive. The average distance hauled is 1.2 miles, with a drop of 600 feet in that distance. Some short grades are as steep as 18-per cent.

At the mine the stope ore is loaded into the trucks from chutes. The ore is fine and soft and runs freely. Some ore from development work is placed in flat, shallow bins beside the truck road, from which the truck driver shovels it into the truck. This is a small part of the total and the contractor is not paid higher rates for it. At the mill, the ore is weighed by the driver on wagon scales and dumped into a 12-ton hopper over the coarse crusher; if the crusher bin is full, the ore may have to be shoveled by hand into a 300-ton storage bin. The chief costs are gasoline, oil, and tires. The contract rate is $0.75 per ton, or $0.90 if the driver has to shovel into the mill bin. This hauling was formerly done with horses and wagons, at 90 cents per ton, and the contractor just made wages and expenses, whereas at present he is making a fair profit on his investment. Moreover, the tight steel truck bodies have eliminated the appreciable loss which formerly resulted from ore rolling off or sifting through the wagon bodies. The trucking contractor also hauls all mine supplies to the mine at a flat monthly rate.

Estimates have been made for a tramway which showed that the present costs could not be lowered on the present scale of operations. If a larger tonnage were handled and if the ore could be collected at the upper end without too much handling, a tramway might be economical. A tramway would also involve changes in methods of control, and in the recording of grades and amounts of contract ore, with probable greater expense for this supervision

WAGE AND CONTRACT SYSTEM

The labor is about 85 per cent native-born Spanish-American and the turnover is very small. At the start of mining operations, all work was on a day's pay basis. The rates were $4.50 for miners, $4.00 for muckers and $5.50 for blacksmiths. These rates have remained unchanged up to the present, but the amount of work done in the mine on day wage is now not over 5 per cent of the total labor, including supervision. In addition to the mine foreman there are two compressormen-blacksmiths, one carpenter-powderman, and one extra man who serves as roustabout and substitute miner - a total of five men per day on company time.

Contracting was first applied to development work. Development contracts include drilling and breaking, mucking, tramming, and installing track and pipe. The contractor furnishes the labor and explosives, including fuse and caps. The company supplies all equipment and other material. Timbering drifts is paid for extra, by the set.

Stope contracting was begun about a year after mining started. It was first put on the basis of pounds of molybdenum sulphide produced. A contract price of 5 cents per pound was paid. The experiment was tried on only one stope and soon proved the method wrong. The natural tendency of the men was to concentrate on high grade ore to the detriment of complete extraction and eventual stoping efficiency. Moreover, it was very difficult to control. The method then tried, which has continued to the present, was to contract stoping on a tonnage basis. The contractors furnish their own powder, and drill, break, sort, shovel, tram, and load the ore into the trucking bins. The mine foreman exercises close control over mining and sorting to insure complete extraction and proper mill feed. The miners are forced to extract ore from the thin spots, or pinches, as well as the swells. The price is such as to enable them to do this, otherwise the use of stoping as a form of exploration would be impossible. Their contract also includes doing all stope timbering and filling.

All contracts are verbal only and are for indefinite periods and amounts of work. Depending on the men available and their desires, the contract may include one or two men of a crew. A drift contract usually includes only the miner. The mucker in that event is paid by the contractor through the company and at company rates; the company does not concern itself with bonuses or payments to contractors' men at rates over the base rates. If the company furnishes a man to the contractor for a few shifts, the man's wages are deducted from contract payments. Ore produced by development contractors is paid for at te rate of $1 per car. In stopes, two men will often comprise a crew, both being contractors on an equal basis. Additional men may be employed by stope contractors.

Contract rates are very seldom changed, as this increases the confidence of the men in their being allowed to make good profits, and their work is therefore more efficient.

Contract Rates

Drifting, machine work	$ 4.00 to 5.00 per foot
Drifting, hand work	5.00 to 5.50 per foot
Raising, machine work	[1]$ 4.00 per foot
Raising, hand work	[1] 5.00 per foot
Timbering, drift	2.00 per set
Ore, development	[2] 1.00 per car
Ore, stoping, hand work	[2] 3.00 per car
Ore, stoping, machine work	[2] 2.50 per car

1 Timbered raise prices are split, part for advance and part for timbering.

2 Cars of ore are figures from truckload weights on the basis of 1,200 pounds per car.

Contractors handle their own supplies from the surface into the mine.

The powderman keeps tally of powder, fuse, and caps issued. The powderman also acts as mine carpenter and frames all standard timber for drift and raise sets. The miners cut their own stulls.

The mine foreman is in direct charge of the men without assistance. The contract system relieves him of the responsigility of forcing production, but at the same time increases his difficulty in maintaining grade and securing complete and orderly extraction.

SAFETY, DRAINAGE, AND VENTILATION

The mine foreman is responsible for accident prevention and administering first aid; a very good accident record has been established. Falling rocks are the cause of the most serious accidents, therefore emphasis is placed on the necessity for barring down and timbering. The nearest hospitals are at Raton, N. Mex., and at Alamosa, Colo. Doctors are available at several towns within a few miles.

The mine makes little water; drainage is by gravity flow out of the adits.

Ventilation is natural and is assisted by raises from level to level at necessary points. In winter, doors are used to check the too strong currents of cold air.

Table 1.- Development costs per foot, by machine drilling June to November (inclusive), 1930

Description: 2,602 feet drifting, 4½ by 6½ in section, 10 to 20 per cent timbered; 475 feet of raising, 5 by 5 or 5 by 10 section, stulled, raw, or cribbed. Total 3,077 feet. Soft ground, but solid.

	Labor	Material	Total
Drilling	$1.00	–	$1.00
Mucking and tramming (including track and pipe)	1.75	–	1.75
Timbering	.16	$0.14	.30
Explosives (including fuse, caps)	–	1.25	1.25
Rails	–	.22	.22
Ties	–	.08	.07
Pipe	–	.23	.23
Drill repair, hose, oil	.05	.35	.40
Steel sharpening	.23	.10	.33
Compressed air	.32	.65	.97
Supervision	.28	–	.28
Total	$3.79	$3.02	$6.81
Compressor and drills, interest and depreciation			.60
			$7.41

Tagle 2.- Development costs per foot, by hand drilling June to November (inclusive), 1930

Description: 959 feet drift; 477 feet raising; total 1,436 feet. Same description as for machine work, but generally softer ground.

	Labor	Material	Total
Drilling	$3.20	-	$3.20
Mucking and tramming	1.60	-	1.60
Explosives	-	$0.70	.70
Track	-	.15	.15
Steel sharpening	.23	-	.23
Supervision	.28	-	.28
Timbering	.16	.14	.30
Total	$5.47	$0.99	$6.46

Table 3.- Mining and development costs per ton[1]

Period	1929	1930 (first 10 months)	1929 and 1930 (22 months)
Tonnage	7,846.5	10,112.3	17,958.8
Development, cost per ton[2]	$ 7.48	$ 5.03	$ 6.10
Mining, cost per ton[2]	$ 5.22	$ 5.74	$ 5.51
Total, cost per ton	$12.70	$10.77	$11.61
Labor, cost per ton	$ 9.78	$ 8.34	$ 8.98
Material, cost per ton	$ 2.92	$ 2.43	$ 2.63
Labor, per cent of total cost	76.9	77.4	77.3
Tons per man shift	.551	.626	.591
Feet of development work per ton	1.05	.71	.86

1 These costs include supervision, and surface expense directly chargeable to mining.

2 The split between mining and development is estimated.

Table 4.- Mining costs in units of labor, supplies, and power, June to November (inclusive), 1930. 6,169 tons

Labor, all mine underground and surface, excluding office	1.47 shifts (8-hour) per ton
Supervision	.05 shifts (8-hour) per ton
Explosives, 40 per cent strength gelatin	2.98 pounds per ton
Power for compressed air	16.9 kw. h. per ton
Timber (average 5-inch round)	4.86 linear feet per ton

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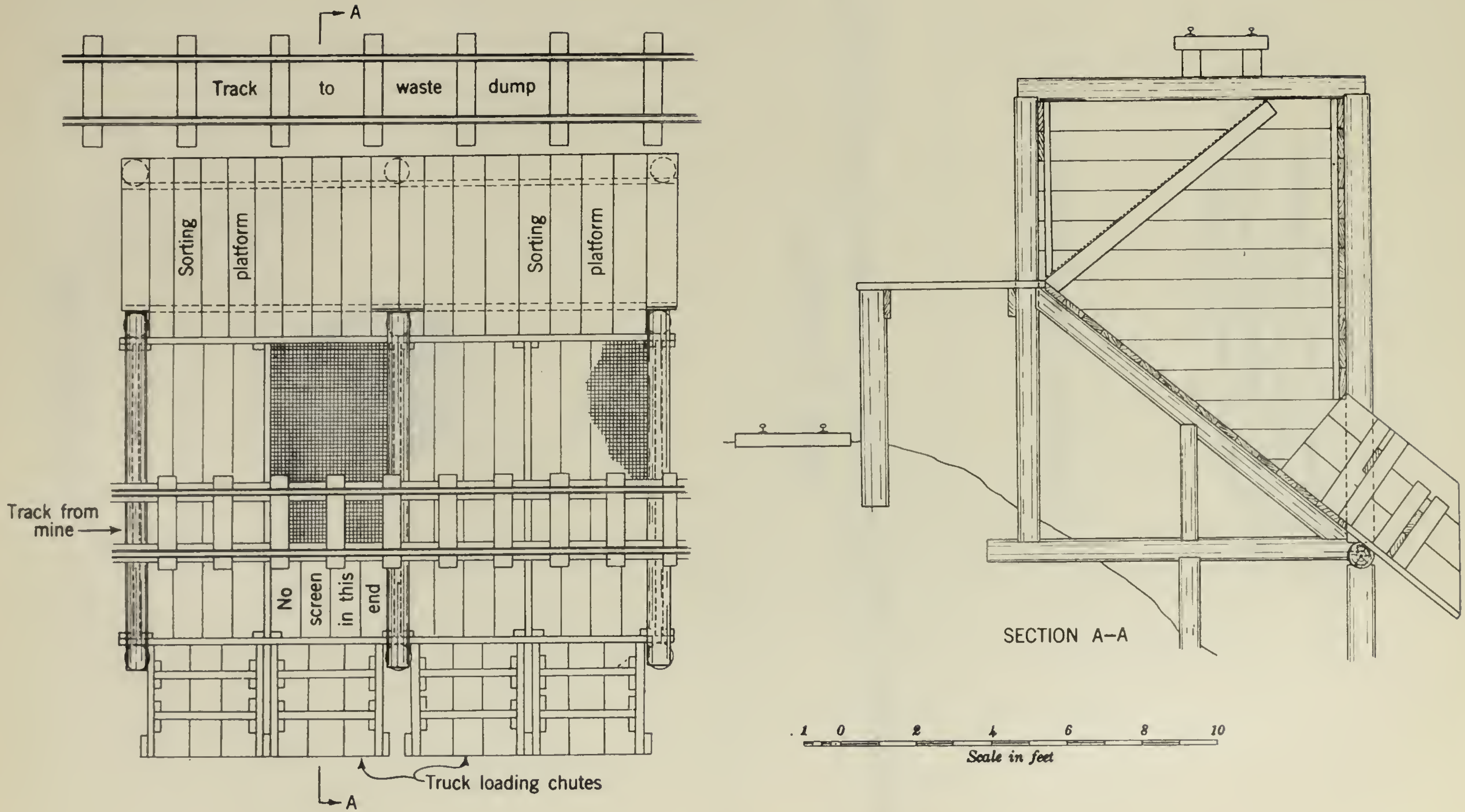

Figure 5.—Typical ore bin, with sorting platform

I. C. 6515 September, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE CHAMPION COPPER MINE, PAINESDALE, MICH.

BY

Albert Mendelsohn

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE CHAMPION COPPER MINE, PAINESDALE, MICH.[1]

By Albert Mendelsohn[2]

INTRODUCTION

This paper, describing the mining practice at the Champion mine of the Copper Range Copper Mining Co., is one of a series of papers on mining methods and costs being prepared by the United States Bureau of Mines.

Mining practice at the Champion mine has undergone considerable change during the last four years. Previous to that time the mine was worked by the horizontal cut-and-fill method, the stopes starting near the shafts and advancing away from them (fig. 2). The floor pillars were mined by the inclined cut-and-fill method, working on the retreat or toward the shafts (fig. 1). At present the mine is being developed to stope entirely on the retreat by a sublevel inclined cut-and-fill method.

ACKNOWLEDGMENT

The author is indebted to the Champion mine organization for much of the detailed information given in this paper.

HISTORY

The Baltic lode was discovered on what is now the Champion property by Dr. L. L. Hubbard in May, 1899. The Champion Copper Co. was incorporated in December, 1899, for the purpose of exploiting the Baltic lode on the Champion property. During 1899, 1900, and 1901. developing and prospecting were done, and in 1902 the first ore was shipped. It was not until about 1909 that the mine reached what might be considered a "balanced" condition, with shaft sinking, drifting, and ore reserves developed well ahead of production.

GEOLOGY

A recent and very thorough survey of the geology of the district and of this mine is contained in Professional Paper 144, The Copper Deposits of Michigan, by B. S. Butler and W. S. Burbank, published by the United States Geological Survey in 1929. The following brief remarks on the geology of the Champion mine are made only to give the reader a summary of the physical conditions in the mine which have affected the mining practice.

At the Champion mine the copper is found in the upper bubble-studded, brecciated capping of a lava flow about 190 feet thick. Almost all the mines in the district find their

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6515."

2 One of the consulting engineers, U. S. Bureau of Mines, and general superintendent, The Copper Range Co.

copper in the top of similar flows. These flows have been tilted so that they dip at angles of 30 to 70°. As a rule, the lodes are unusually straight for very long distances on the strike. The hanging wall of a lode is the bottom of the next succeeding lava flow. The footwall of a lode is the trap lying under the amygduloidal, bubble-studded top of the flow. In some places the lodes outcrop; in other places they are covered with heavy glacial drift.

BALTIC LODE

The lode being mined at the Champion mine is the heavily brecciated top of what is known in the district as the Baltic flow. The brecciated portion of the lode ranges from 6 to 50 feet in width. Under this breccia is the thinner, tighter, amygduloidal portion of the lode; under the amygduloidal portion is the trap. Native copper is found deposited in the rock fragments of the breccia, in the cementing material of the breccia, in the amygdules, and in the trappy rock under the amygduloidal part of the lode; in some places masses of copper many tons in weight extend far into the foot trap of the lode.

The Baltic lode on Champion property is 8,000 feet in length, is very straight along the strike, and dips uniformly at 70° to the west for 3,000 feet in depth, at which point the dip begins to flatten slightly. The width of the copper-bearing portion of the lode ranges from 10 feet to as much as 80 feet in a few places. The average width now mined is 17 feet.

The lode rock is hard; the standard 1-man drilling machines used in the mine drill 10 inches to 14 inches a minute, unless mass copper is encountered. The hanging wall, which is the bottom of another flow and is of a blocky structure due to quick cooling, is full of minute seams running in apparently every direction, with larger seams running parallel to the strike and dip. When exposed over an area of only a few square yards and unsupported by timber or fill the hanging wall will begin to disintegrate and fall into the stope. There is no definite foot slip; the lode becomes more and more trappy on the foot side the deeper it is followed into the flow. The wide portions of the lode seem to be due to large accumulations of fragmental material on the very top of the flow. In some places the lode is split into two branches separated by very trappy, amygduloidal rock.

Copper is not distributed uniformly in the lode. In a large way it follows chutes pitching from the north down toward the south. The copper may occur on the hanging side, in the middle, or on the foot side of the lode, or it may be distributed from hanging to foot. In a smaller way copper occurs in patches surrounded by barren lode rock. It is very patchy along the strike of the lode and across the lode, so that the absence of copper in any drift opening in one of the chutes may mean only that the copper lies to the foot or hanging of the opening, or that it may lie just ahead of the opening.

There are many cross slips or seams running parallel to the strike but dipping at nearly 90°. Even the larger cross slips do not throw the lode more than a few feet. The southern boundary of the Champion Mine is a fractured zone of considerable extent.

The rock shipped from the Champion mine for the last five months of 1930 ran 46 pounds of refined copper per ton. Before hand-sorting the rock broken in stopes and drifts ran 25 pounds per ton. It is estimated that before selection of stoping ground all the lode rock opened by development would have produced only 9 pounds per ton.

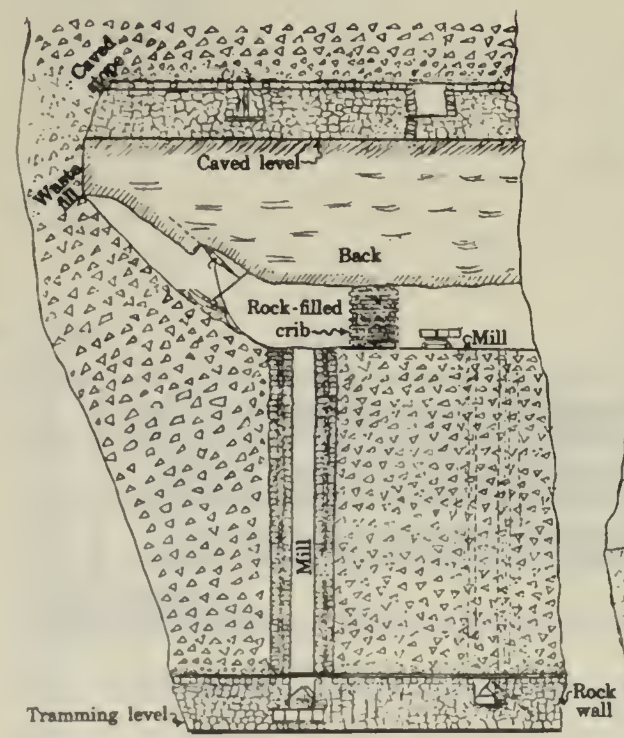


Figure 1.–Method of extracting floor pillar, old stoping system

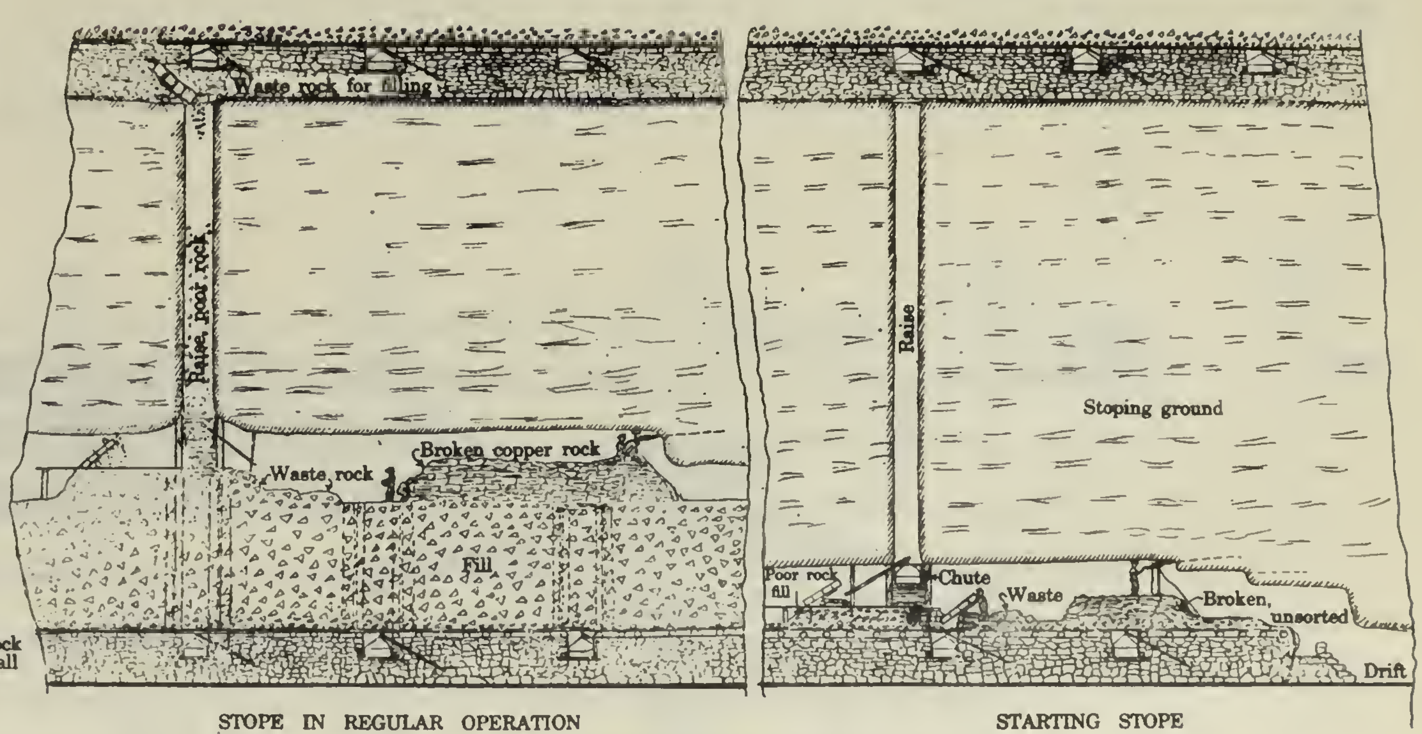


Figure 2.–Old stoping system

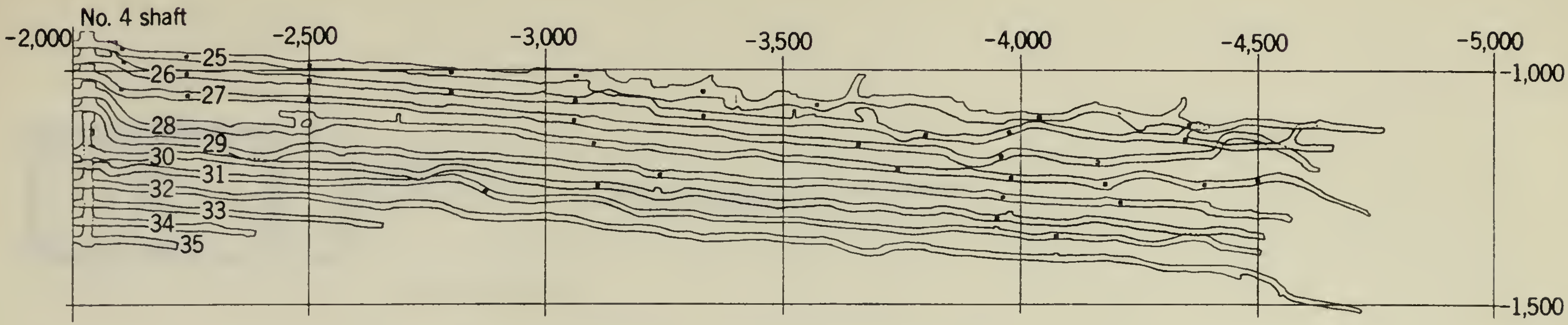


Figure 3.-Plan of drifts at southern end of Champion mine

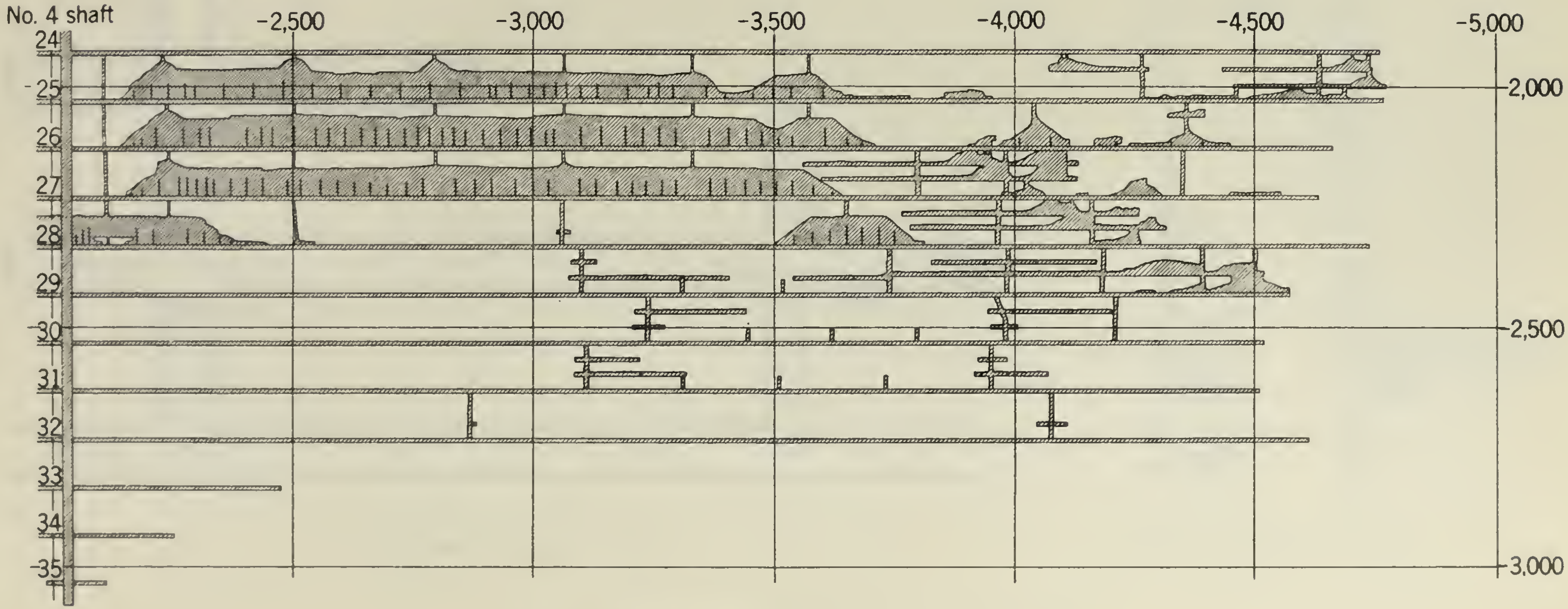
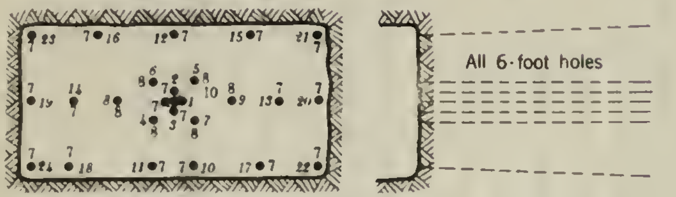


Figure 4.-Stoped areas at southern end of Champion mine



NOTE: Figures indicate number of sticks of dynamite:
Italic numbers the order of firing

Figure 5a.—Drill round

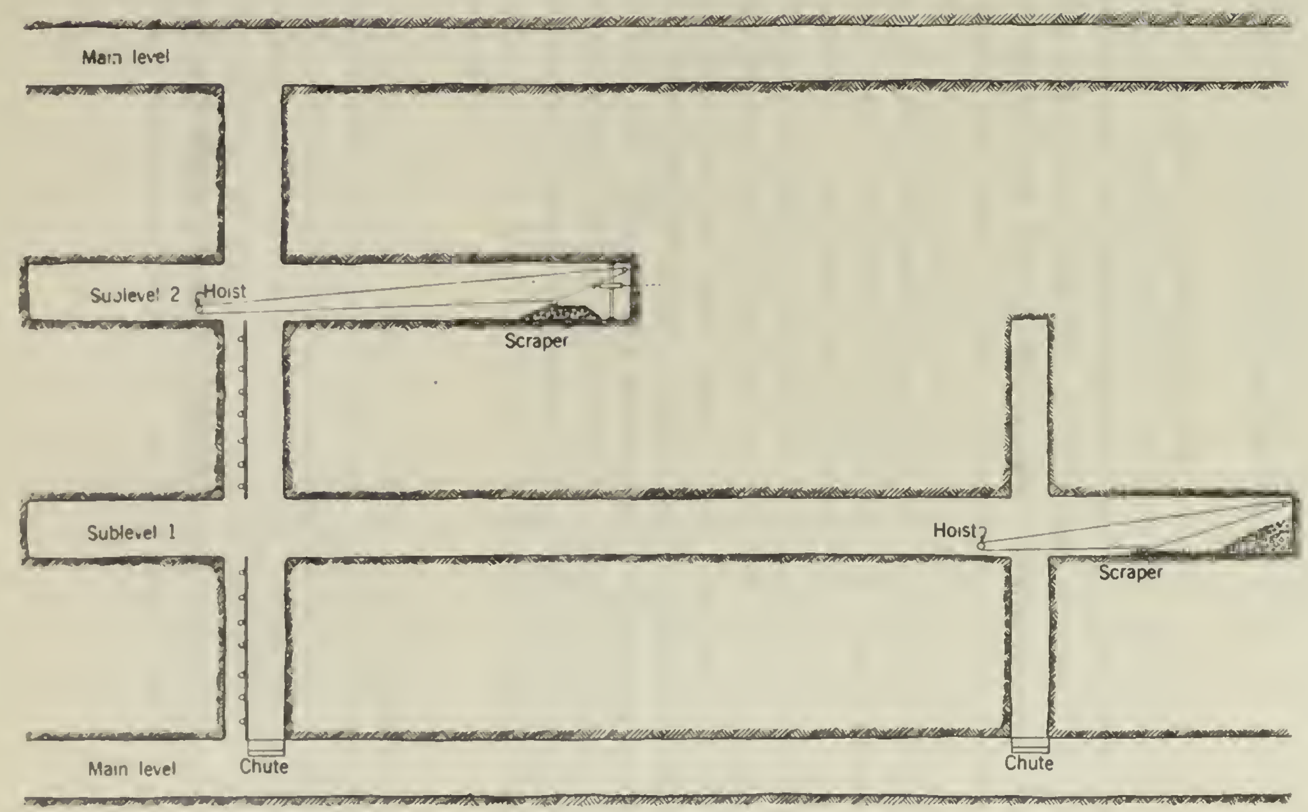


Figure 5.—Development of sublevels and raises

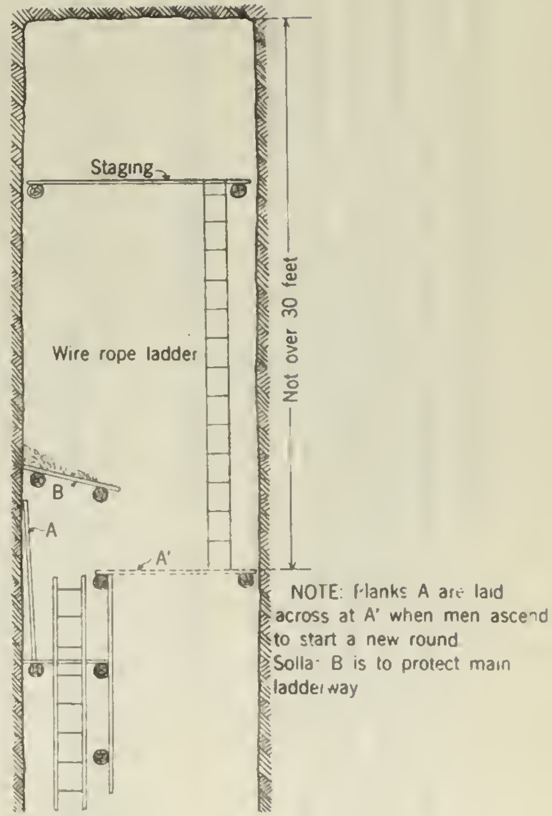


Figure 5b.—Top of 4 by 8 foot raise

METHODS OF PROSPECTING AND EXPLORATION

Where the glacial drift is not too heavy prospecting is generally done by test pitting. A large amount of diamond-drill exploration has been done in the district. Within the mines exploration is done by crosscuts and drifts and by diamond drilling. Geophysical prospecting methods have been successfully used to follow the major geological formations in districts covered by heavy glacial drift.

METHODS OF SAMPLING AND ESTIMATE OF TONNAGES AND VALUES

At none of the mines operating in the copper country of Michigan is sampling done. It has been attempted in a few places but proved too costly, and the results were not dependable.

Tonnages and values are estimated by the mining captains or engineers, or both, who designate the monthly openings as "good", "fair", or "poor". The decision as to what ground can be mined profitably is made by the superintendent and mining captains, who rely on their experience in judging the ground.

METHODS OF DEVELOPMENT AND MINING

The Champion mine is developed by four inclined shafts, three of which are in operation at present. The shafts were started on the lode on a dip of about 70°. Thus far this dip angle has been satisfactory, but at present depths the lode is beginning to flatten. Main levels are driven from the shafts at intervals of 100 feet on the dip. These main levels are driven a convenient width (from 9 feet to 13 feet), depending on the character of the ground with respect to copper and "loose" ground. These main levels follow the copper in so far as is possible but avoid sharp curves such as would make for inefficient haulage. With a lode as wide as the Baltic, and the copper occurring as it does on either the foot side or hanging, or both, supervision of these main levels drifts becomes very important. These drifts explore as well as develop. In former years, before the ground was under stress these main-level drifts were driven with speed as the second consideration, the chief concern being to develop the copper-bearing portion of the lode to its full width as the drift advanced. Such a scheme gave assurance that the level, when finally walled or timbered, would be under the copper, and stoping could proceed in an orderly manner without further exploration. With the decision to mine "on the retreat" the main-level drifts were given a minimum width for the sake of reducing maintenance costs and increasing speed of advance, but at the same time they are closely watched and allowed to meander in an effort to keep in the best part of the lode and thus pay their way as driven. Figure 3 shows a plan of the drifts in the southern end of the Champion mine.

Figure 4 shows stoped areas on the higher levels and development for sublevel mining on the bottom levels, and Figure 5 shows a section and details of the mine developed for sublevel mining on the retreat. Raises are approximately 200 feet apart. Subdrifts are driven 33 and 67 feet, respectively, above the main level rails. Subdrifts are driven the full width of the copper-bearing portion of the lode. Speed of drifting is not the first consideration, the chief concern here being to develop and explore thoroughly, so that when stoping starts there need be only a minimum of delay for further exploration.

Shafts

All shafts at the Champion mine are 9 feet by 23 feet, rock measurement, and are sunk on a dip of 70°. The shafts are in trap rock, which is very hard but full of slips and fractures which necessitate complete timbering. Timber consists of dividers, wall plates, sleepers, spare pieces, and studdles of 12 by 12 inch Douglass fir, stringers of 8 by 10 inch fir, and lagging of round, peeled, cedar poles. Sixty-pound rails are used in the shafts.

Skips

The skips have a capacity of 6 tons and have side plates extending up well under the bail, to prevent spillage into the shaft when they are loaded. Figure 6 shows a safety bonnet for the skip, and Figure 7 shows the plat and loading arrangement with the skip in loading position. The dumping arrangement for skips is shown in Figure 8.

Cages

Cages (fig. 9) have two or three decks, holding 12 men on a deck. The upper and middle deck floors and the top of the cage are removable, enabling the cage to be converted quickly into a timber and supply truck.

Shaft Sinking

A shaft-sinking crew consists of four men and one puffer boy on each of the two shifts. These four men drill, blast, muck, and timber. The timber is kept down within 12 feet of where the men are working. Shaft lines are set by the engineers and the head mine timberman.

Figure 10a shows the drill round used at present in shaft sinking. Machines are Gardner-Denver No. 11 sinkers and are used with water.

Fractured ground frequently causes the holes to cave in, and for this reason light machines which can be easily raised or lowered by the miners while drilling are used.

Seven-eighths-inch hexagon, hollow drill steel, collared, with $3\frac{1}{4}$-inch shank, is used. Double Carr cross bits are used, with gages as follows: Starter, 1-7/16 inches; 4 foot, 1-3/8 inches; 6 foot, 1-5/16 inches; 8 foot, $1\frac{1}{4}$ inches. Drill steel is handled in and out of the mine in bundles. Blasting is done with fuse and cap, using 1-inch Giant Gelatine, 40 per cent powder.

Mucking is being done at present by hand into 1000-pound capacity buckets; see Figure 10b, which shows the shaft-sinking method. These buckets are dumped automatically into a flop pan controlled by the puffer boy, the rock running into a bin of 30 tons capacity. Loading from the bin into the skip is done by means of a sliding chute. The bucket carriage runs on an 8-inch I beam.

Present practice is to sink 300 feet before letting out the main hoisting ropes to bring the skip down. A rock pentice 8 feet thick, is left under the skip pit to protect the men engaged in sinking operations. When 300 feet of sinking has been completed stringers are put in and the level stations cut. Then the pentice is blasted out, and the skip roads are connected.

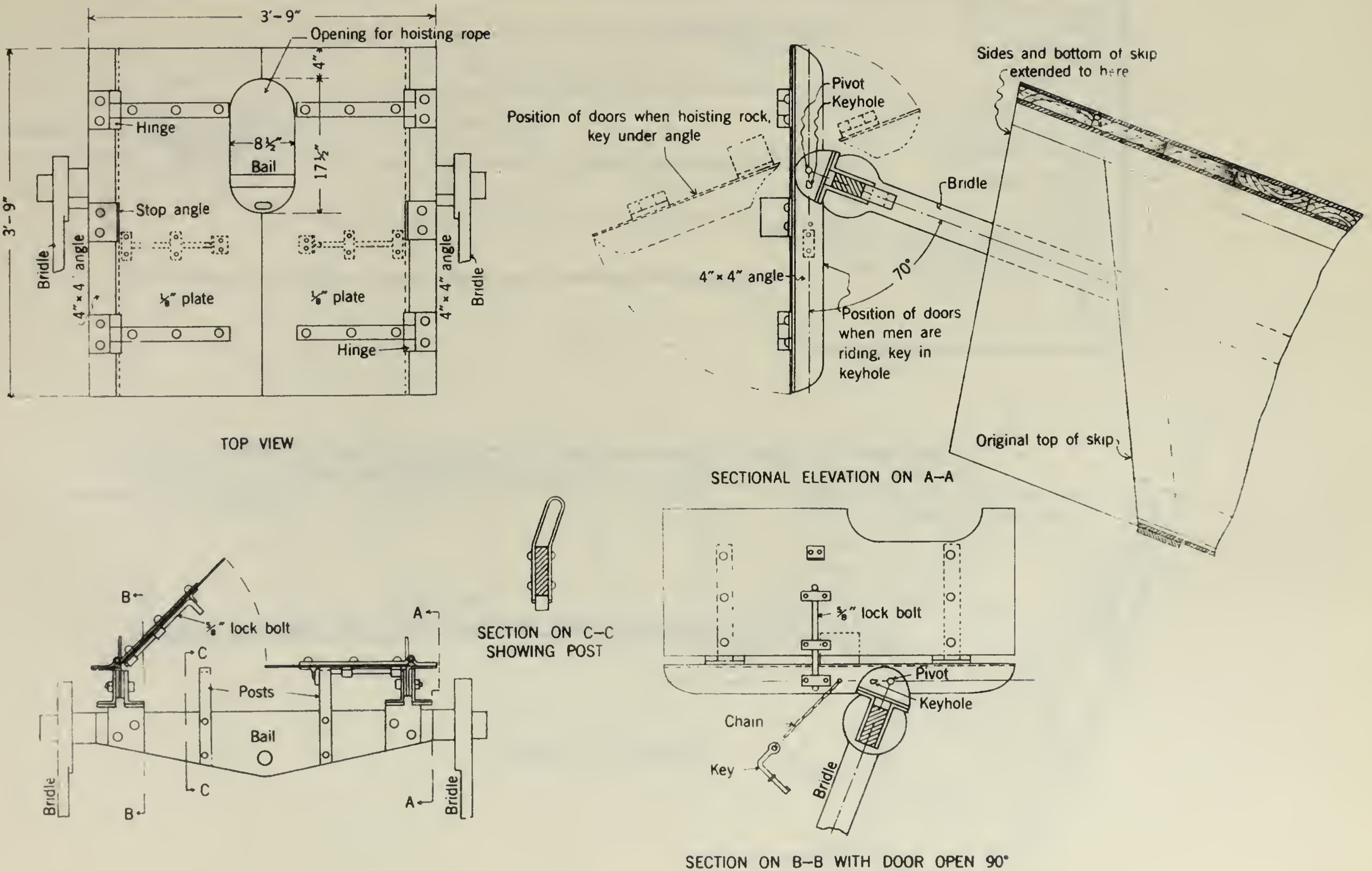


Figure 6.—Safety bonnet for skip

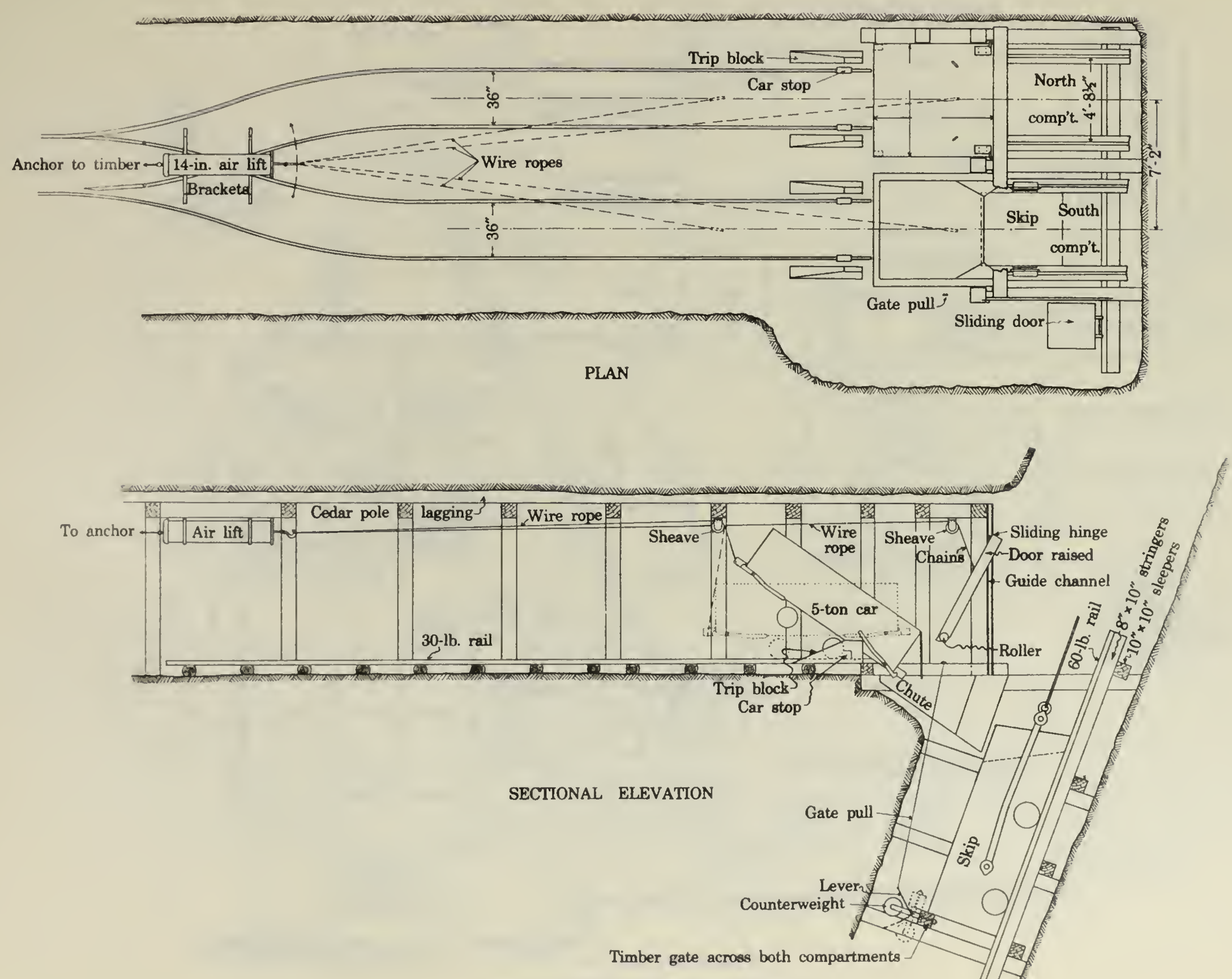


Figure 7.–Plat and loading arrangement

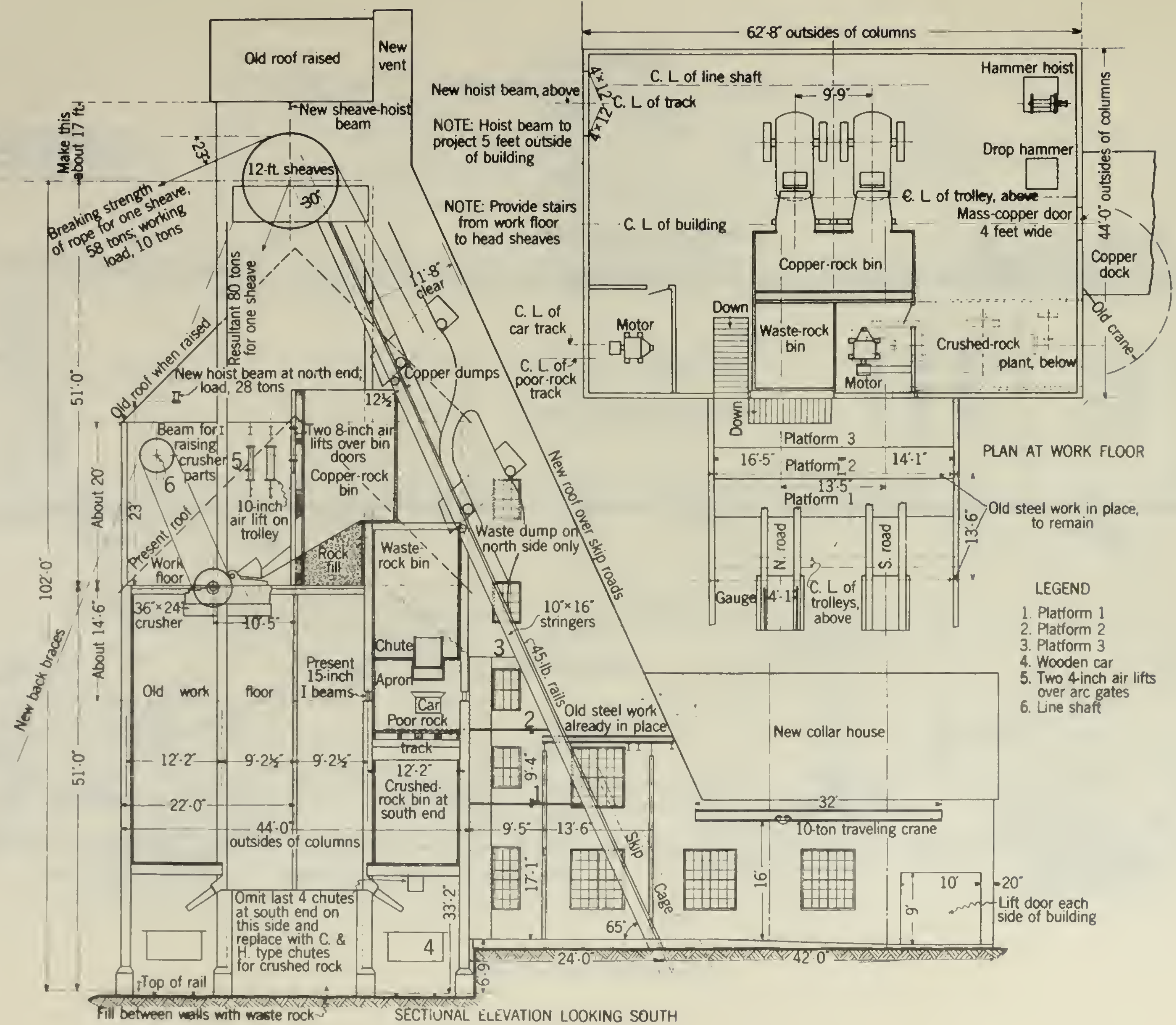


Figure 8.–Skip-dumping arrangement in rock house

BILL OF MATERIAL

Mark	Required	Size	Length	
1	2	8" [20 lb/ft.	21'0"	Sill
11	2	6" [7.25 lb/ft.	21'0"	Top stringer
2	2	4×4×1/2" L	21'0"	Bottom angle
3	2	4×4×3/8" L	3'7"	Lower end angle
4	1	Do.	3'[illegible]"	Do.
5	2	Do.	4'3"	Lower end angle
6	1	Do.	3'2"	Do.
7	2	Do.	3'7"	Deck support
8	2	Do.	3'7"	Do.
9	2	3×3×1/4" L	3'2"	Removable deck
A	2	Do.	4'8"	Brace
B	2	Do.	7'8"	Do.
C	2	Do.	8'0"	Do.
10	2	1/2"×92 1/2" Pl.	10'4"	Bottom plate
12	2	5/8" mesh, 42"	18'6"	Side screening
13	2	1/2"×18" Pl.	4'0"	Side plates
14	2	1/2"×18" Pl.	4'6"	Side plates, lower
15	2	1/2"×49" Pl.	4'10"	Side plates, upper
16	2	1/4"×16 1/2" Pl.	1'5"	Gusset
17	2	1/4"×10" Pl.	1'3"	Do.
18	2	1/4"×12" Pl.	1'8"	Do.
19	2	Do.	1'8"	Do.
20	1	1/2"×[illegible]" Pl.	4'0"	Cover plate
21	1	1/2"×22" Pl.	4'0"	Cover door
22	2	1/2"×10" Pl.	0'10"	Reinforcing plate
23	2	1/2"×10" Pl.	4'0"	
24	2	1/2"×43" Pl.	3'10"	Deck floor
25	2	3/8"×10" Pl.	3'7"	Deck front
26	4			Standard hub
27	4			Standard wheel
28	4			Standard shoe
29	4			Standard guide
30	1			Standard bail
31	2			Standard bridle and draft lug
32	1	1/2"×47" Pl.	4'3"	Lower end plate

NOTE: Wire netting for sides; Tyler's best double crimped iron-wire screen: Diameter of wire 0.1620 inch, mesh ⅝ by ⅝ inch center to center of wires, 42 inches wide: Galvanized after weaving

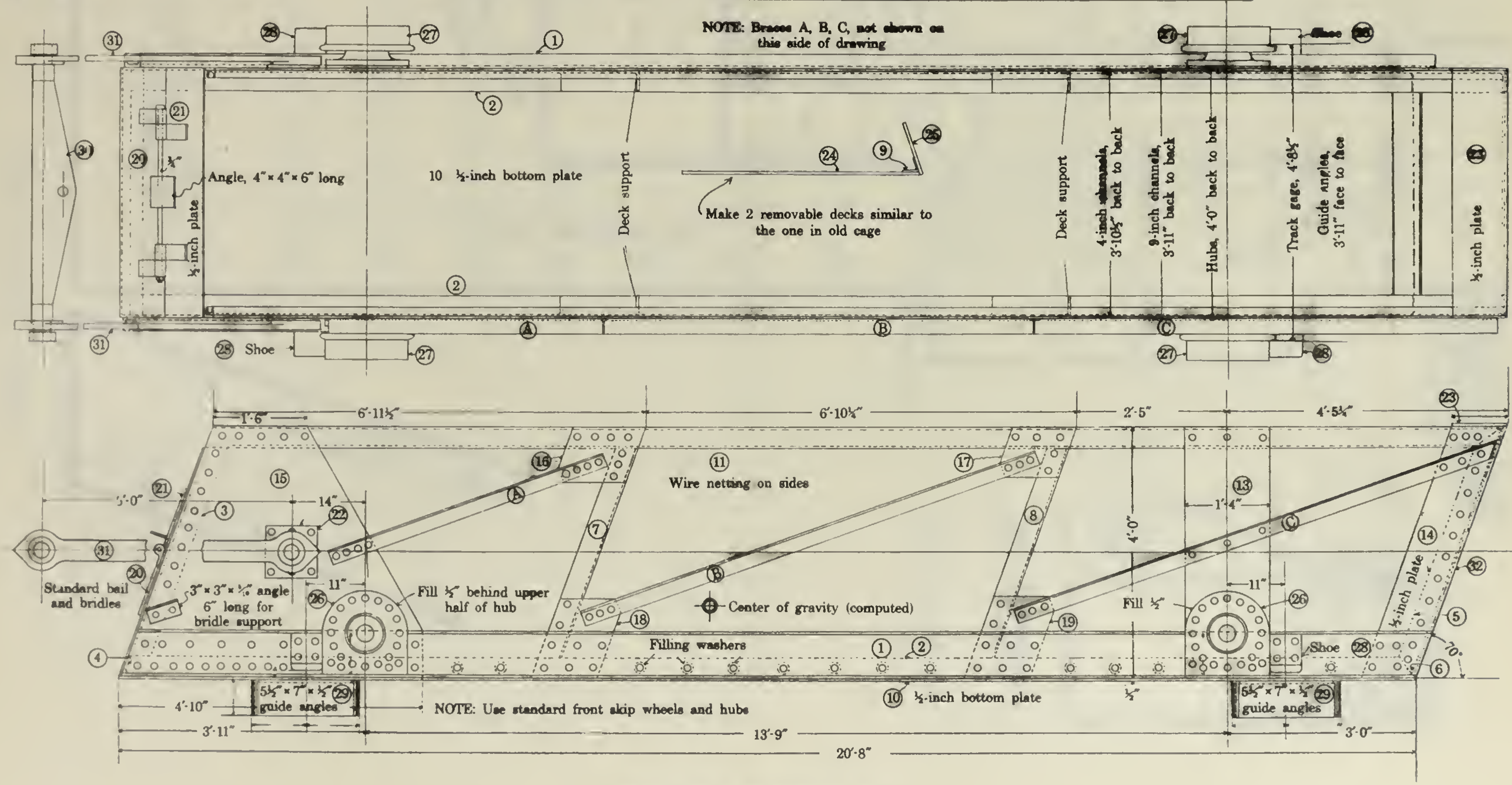


Figure 9.—Design for three-deck man car

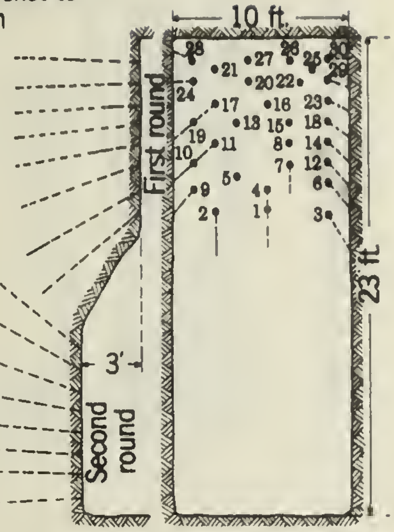
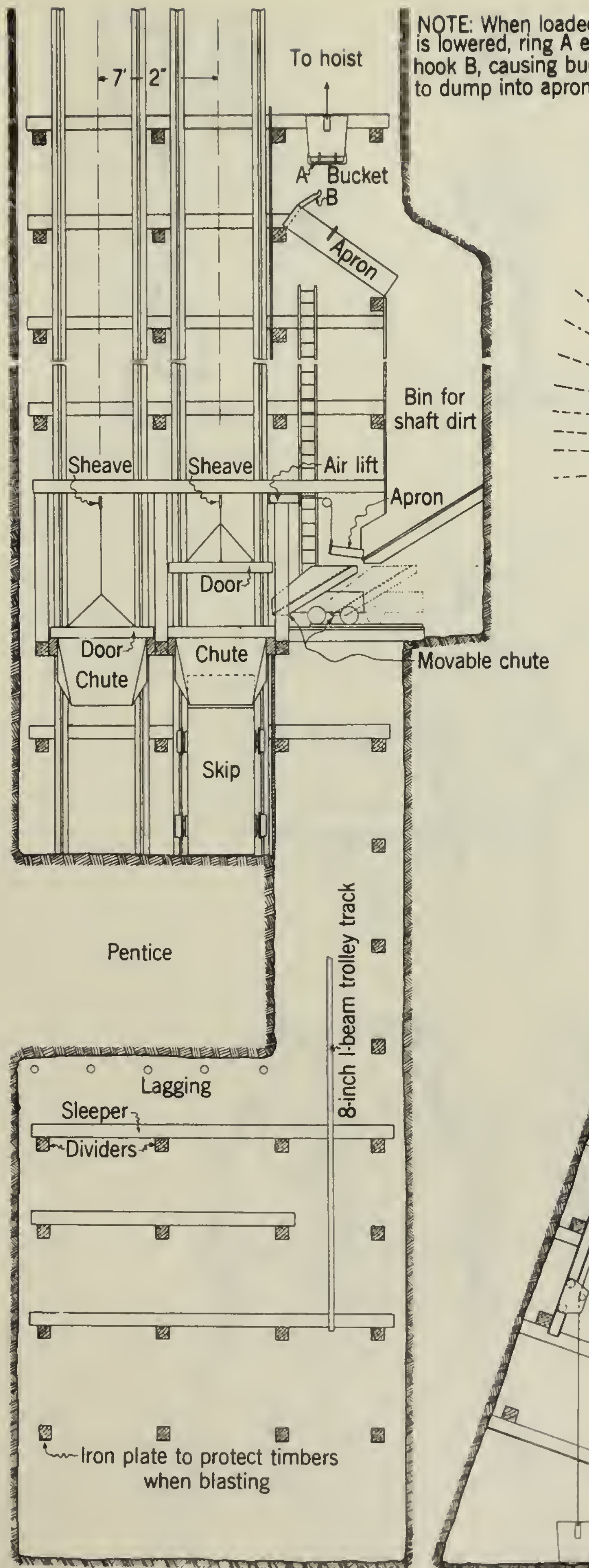


Figure 10a.–Drill round



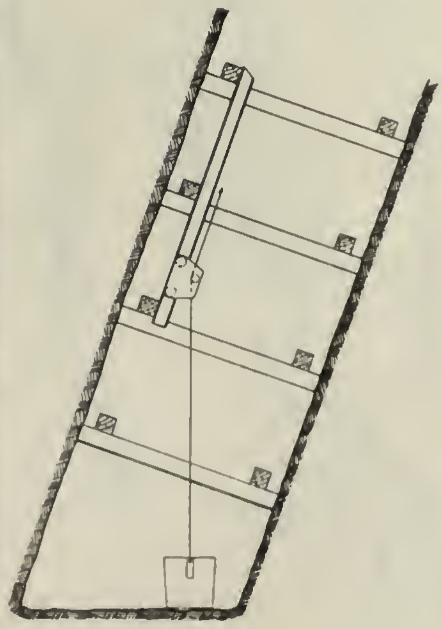


Figure 10b.–Shaft-sinking method

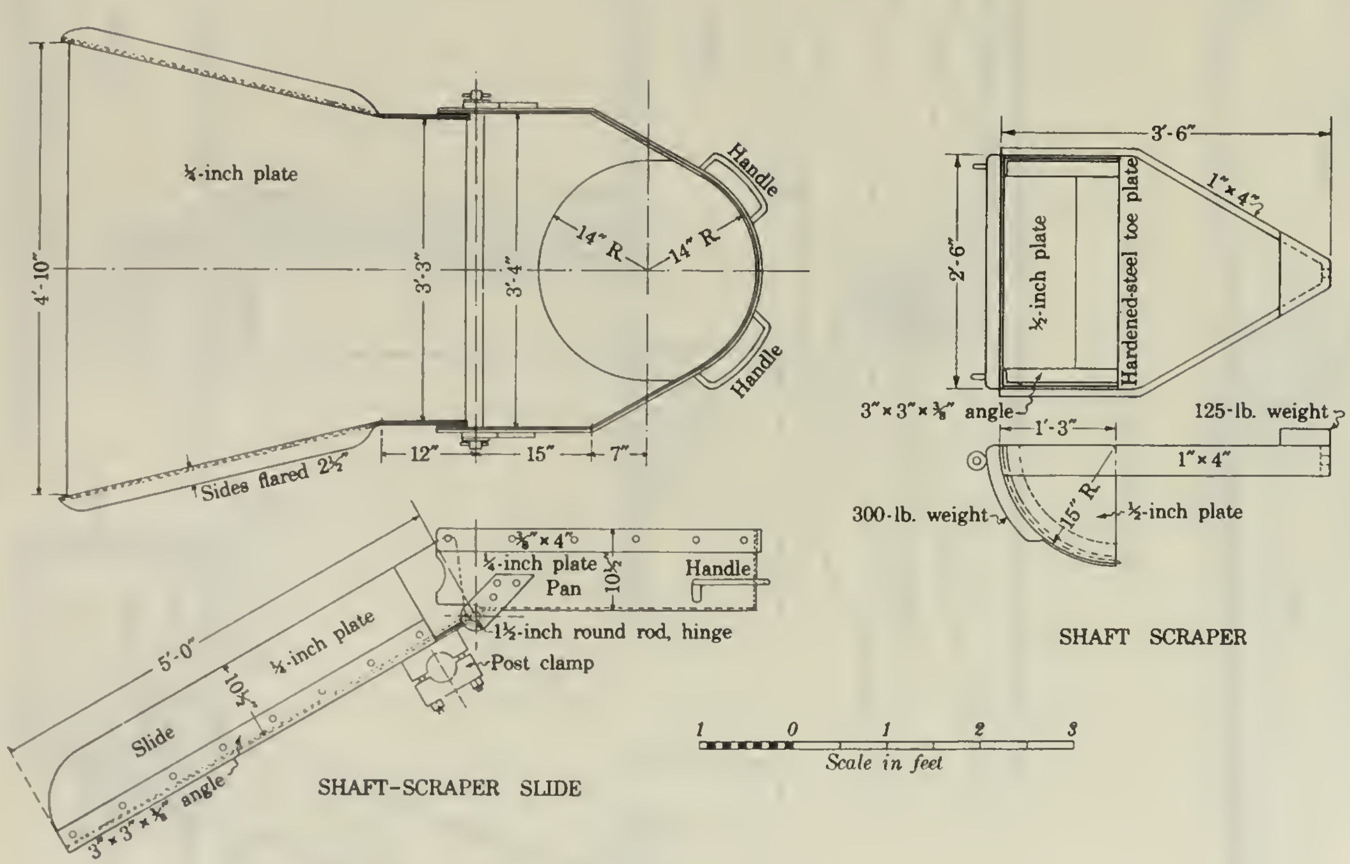


Figure 11.—Design of scraper and slide for mucking in shaft

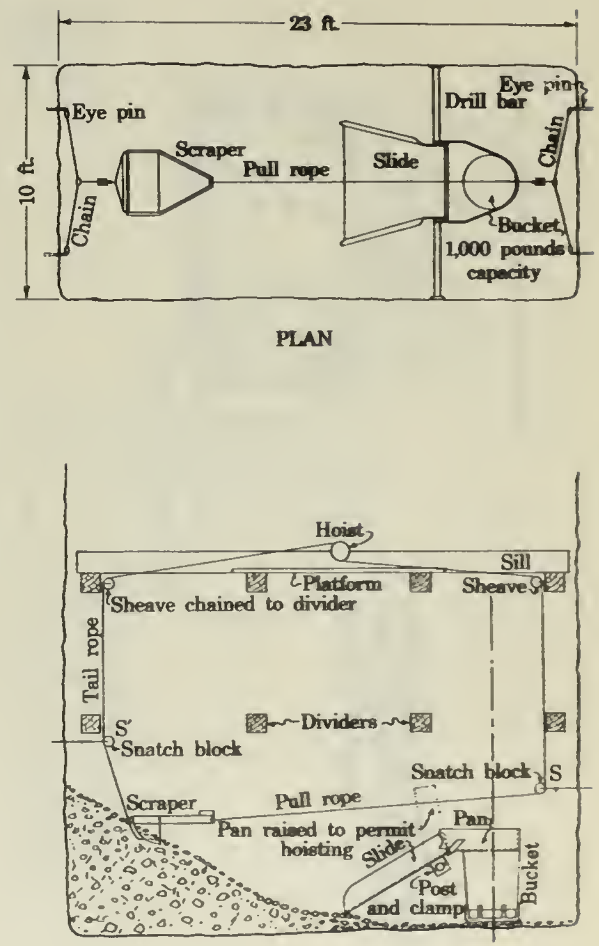


Figure 12.—Scraper mucking at bottom of shaft; general arrangement

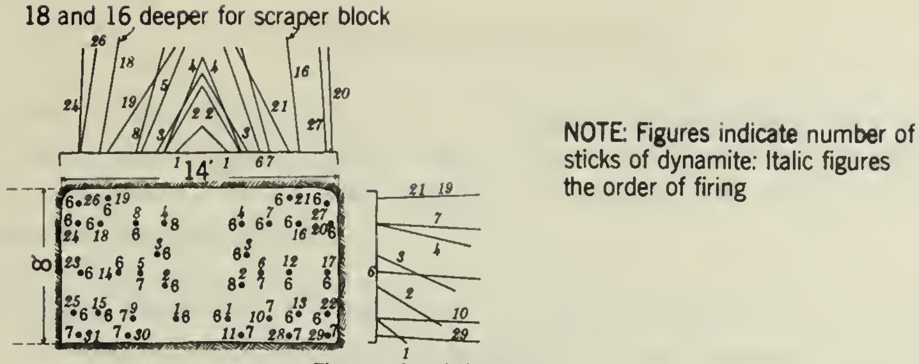


Figure 13a.–Drill round

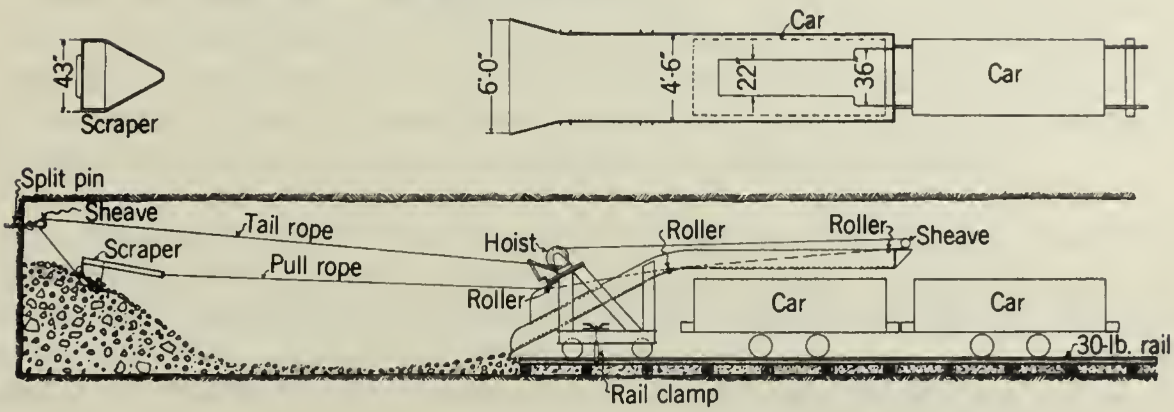


Figure 13.–Drift-mucking method

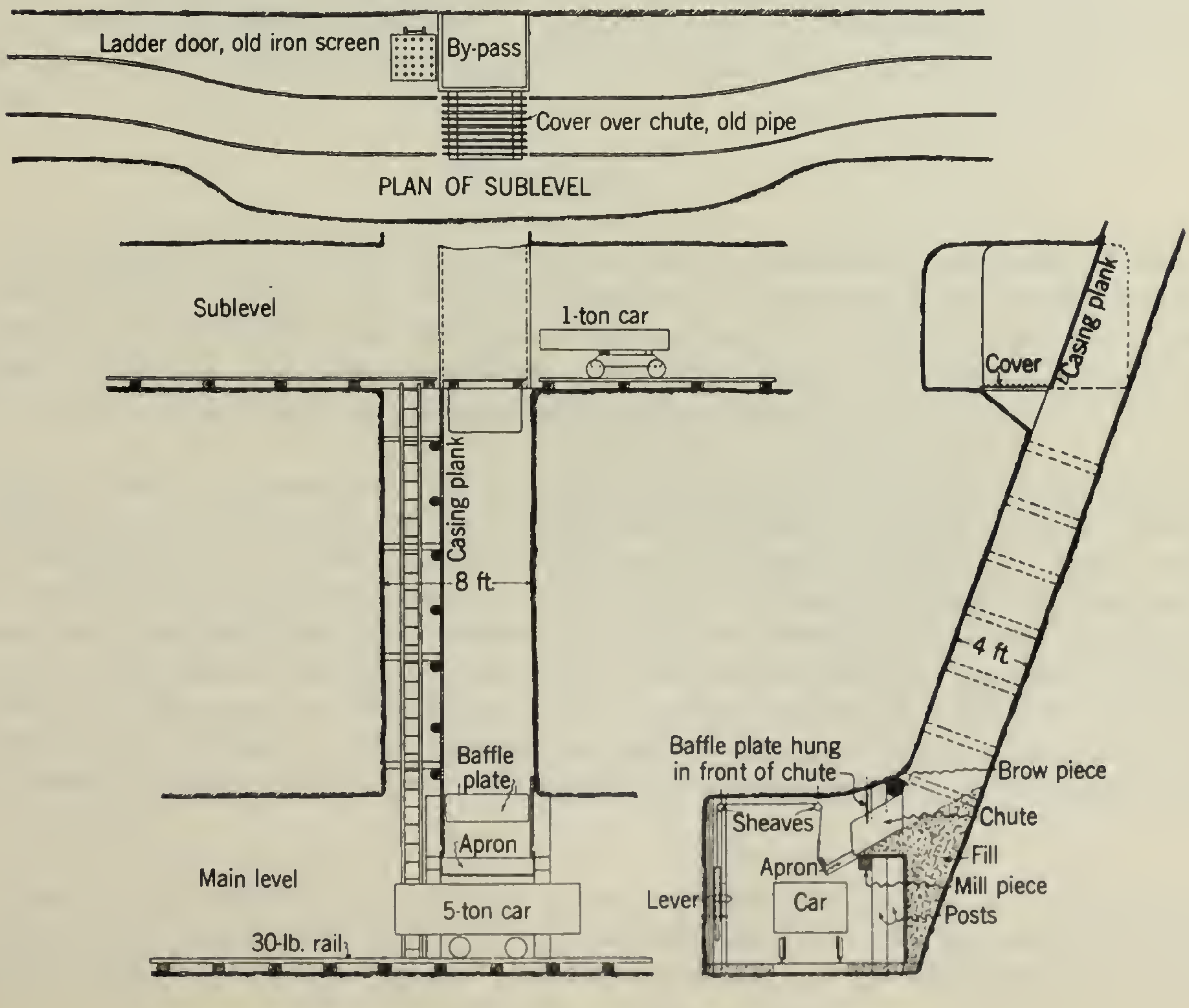


Figure 14.–Raise and chute construction

Shaft Mucking With Scraper

Mucking in the bottom of the shaft by means of a scraper and slide is now being tried. It is yet in an experimental stage, having been used only a few days, but is showing promising results. Figure 11 gives the designs of scraper and slide, and Figure 12 shows the general arrangement in the shaft.

The first operation is to scrape all or most of the dirt to the north end of the shaft; then the slide is set up as low as possible, and the scraper is reversed for loading the bucket. When the south cut is being blasted most of the dirt is thrown north, which reduces the preliminary scraping work. The slide is attached by clamps to a horizontal bar or post. Sheaves (or snatch blocks) S and S' are attached by clevices to chains strung across the ends of the shaft through eye pins. These chains have several rings or links, which permit changing the position of the outfit toward foot or hanging as desired.

When the bucket is full the hinged pan or apron is turned back, and the scraper rope is pulled to one side to permit hoisting. Two good scraper loads will fill the 1,000-pound bucket in about 20 seconds. The total time for a round trip is about 2½ minutes, or about 25 buckets per hour, hoisting a distance of 150 feet. A larger bucket is contemplated. When the mucking is finished slide and scraper are hoisted up and hung out of the way until needed again.

The scraper hoist is mounted on a platform far enough above the bottom to be safe from blasting. Of course, some hand mucking will still have to be done, when sinking through the pentice, before the shaft is widened. As stated above, the scheme is new, and some difficulties will no doubt develop that will have to be ironed out, but in the main it is working very well.

Drifts

Main-level drifts are driven 13 feet wide when in copper rock and about 9 feet wide when in barren rock. The rock is hard and firm, and little support is needed except for loose material in the neighborhood of slips crossing the drift. This loose rock is caught up by props with 8-inch flat timber headings and by occasional drift sets. Mucking in the main level drifts is done with portable scraper slides (fig. 13).

The standard organization for main-level drifts is two miners and one helper per shift on each of the two 8-hour shifts. These men mine, muck, tram, dump their rock into the skip, take care of the slusher engine and slide, and keep the drift in a clean, orderly condition. There are two machines in each drift, rigged on ordinary 3-inch iron-pipe columns, single jack, with 3-inch arms. The machines are Ingersoll-Rand Model RA-72, equipped with tappet chuck, using water. The drill steel is 1-inch hollow hexagon, lugless, with Double Carr cross bits. The gages of the bits are as follows: 2 foot steels, 1-7/16 inches; 4 foot, 1-3/8 inches; 6-foot, 1-5/16 inches; and 8-foot steels, 1-1/4 inches. The round drilled is shown in Figure 13-a. Miners are permitted to blast as soon as the whole round of holes is completed.

The dirt is mucked with scraper slide, as shown in Figure 13. Sullivan Type HDE-4, 15 hp., D.C. electrically driven slusher engines are used. The scraper slide used is that commonly known as the "Osana" type. It has been found that the sharpest curve that can

be driven with this type scraper slide in a drift 10 feet wide is of 35-foot radius. It requires an average of two hours to muck out a round.

Raises

Raises are driven 4 feet by 8 feet, rock measurement. The rock is hard and firm, and little support is needed. However, inasmuch as the raises are to be used later for ladder road and ore pass they are timbered when driven, as shown in Figure 14. The raises are always put up at a point where there is a good showing of copper, and consequently usually pay their way.

Machines used are self-rotating and self-feeding wet stopers. Two miners working together with two machines comprise the raising crew and work only one shift a day. Drills used are 1-inch hollow hexagon. The starter bit is 1-9/16 inches, the second steel 1-1/2 inches, and the third steel 1-7/16 inches. The round drilled is shown in Figure 5a.

When the raise is advanced 30 feet from the level it is timbered as shown in Figure 14, and a chute for handling the muck is built at the bottom. The set of timbers at the top of the ladder road is covered with heavy timber to protect the ladder road when blasting is done. Lagging is omitted from the top set, between the ladder road and the dirt compartment, to allow for ventilation. When the raise is advanced 30 feet above the timber it is again timbered and lagged, as shown in Figures 14 and 5b.

MINING (STOPING)

The method of mining at the Champion mine is being changed from horizontal cut-and-fill working on the advance to sublevel inclined cut-and-fill, working on the retreat.

Old Mining Method

The Baltic mining method, which is well described in Peele's Handbook, is still being used in the upper levels of the Champion mine. Drifts were driven 8 feet high and the full width of the copper bearing part of the lode. When 200 to 500 feet had been drifted in copper ground the drift was enlarged to a height of 14 feet. Dry walls of discarded rock were built on the foot and hanging sides to a height of 7 feet and the top covered with timber and lagging (fig. 15 which is a section through a level, old mining method). Raises were put up from the top of this wall at intervals of about 500 feet, and fill was dropped from the level above and spread behind the foot and hanging dry walls. In the footwall mills (ore passes) were laid out at intervals of 33 feet. Horizontal cuts about 4½ feet thick, were taken off the back. This rock was picked, the copper rock was trammed to the mills in small stope cars, and the rock discarded was thrown to the foot and hanging sides of the stopes. The mills (ore chutes) were then walled up with the discarded rock, and the stope was again filled with waste rock dumped down the raise from the level above. When stoping had begun on the top of the dry walls the drift was again driven until enough ground had been opened for another stope.

Horizontal cut-and-fill stoping continued for the entire length of the level until all but 30 feet under the level above had been mined out. The extraction of this 30-foot floor pillar was made by inclined cut-and-fill stoping, commencing at the inside boundary (figures 1, 2 and 16). Filling for the inclined cut and fill was obtained by wrecking the level above, the last round blasted allowing the fill from above to rush down into the stope.

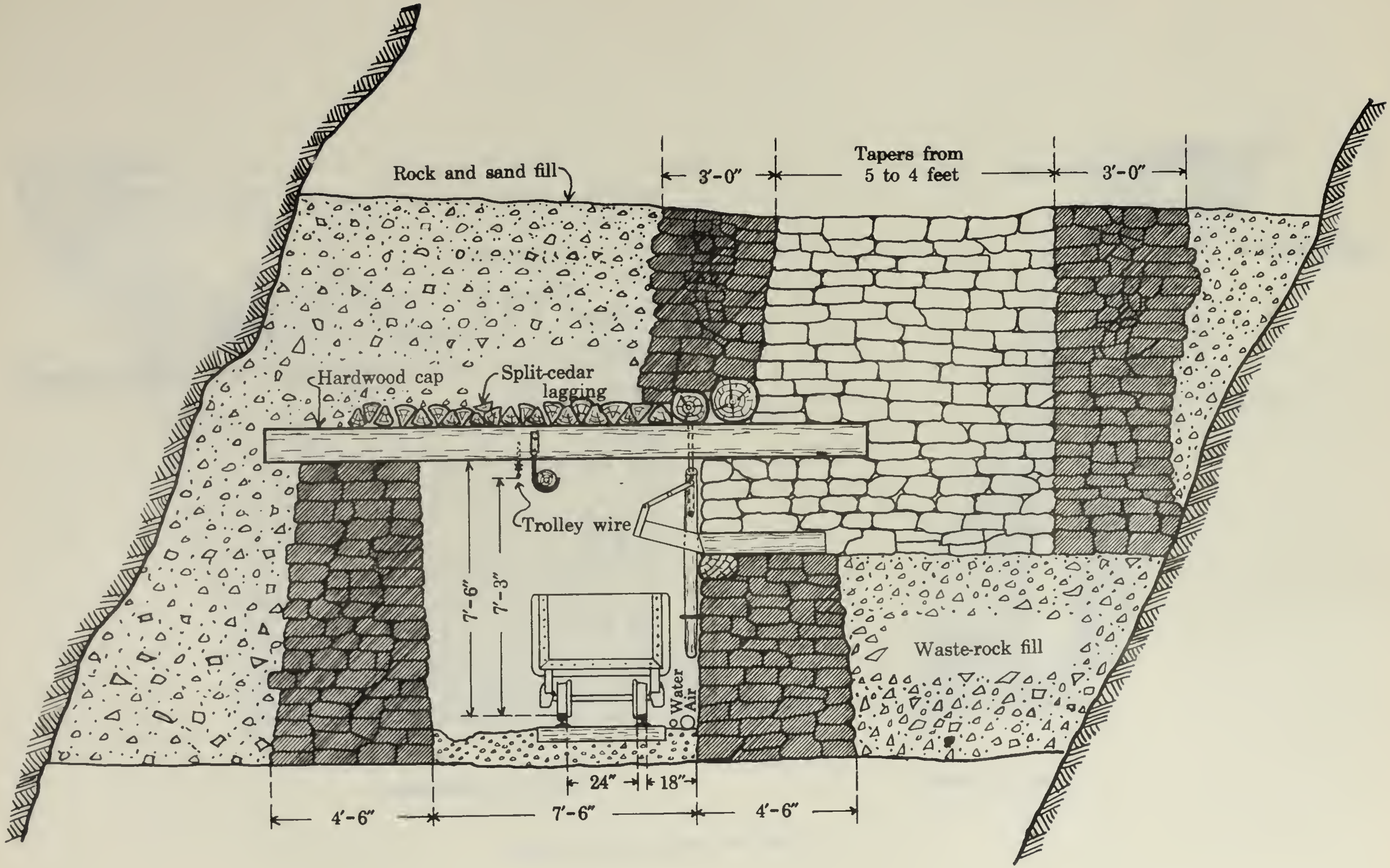


Figure 15.—Section through level, old mining method

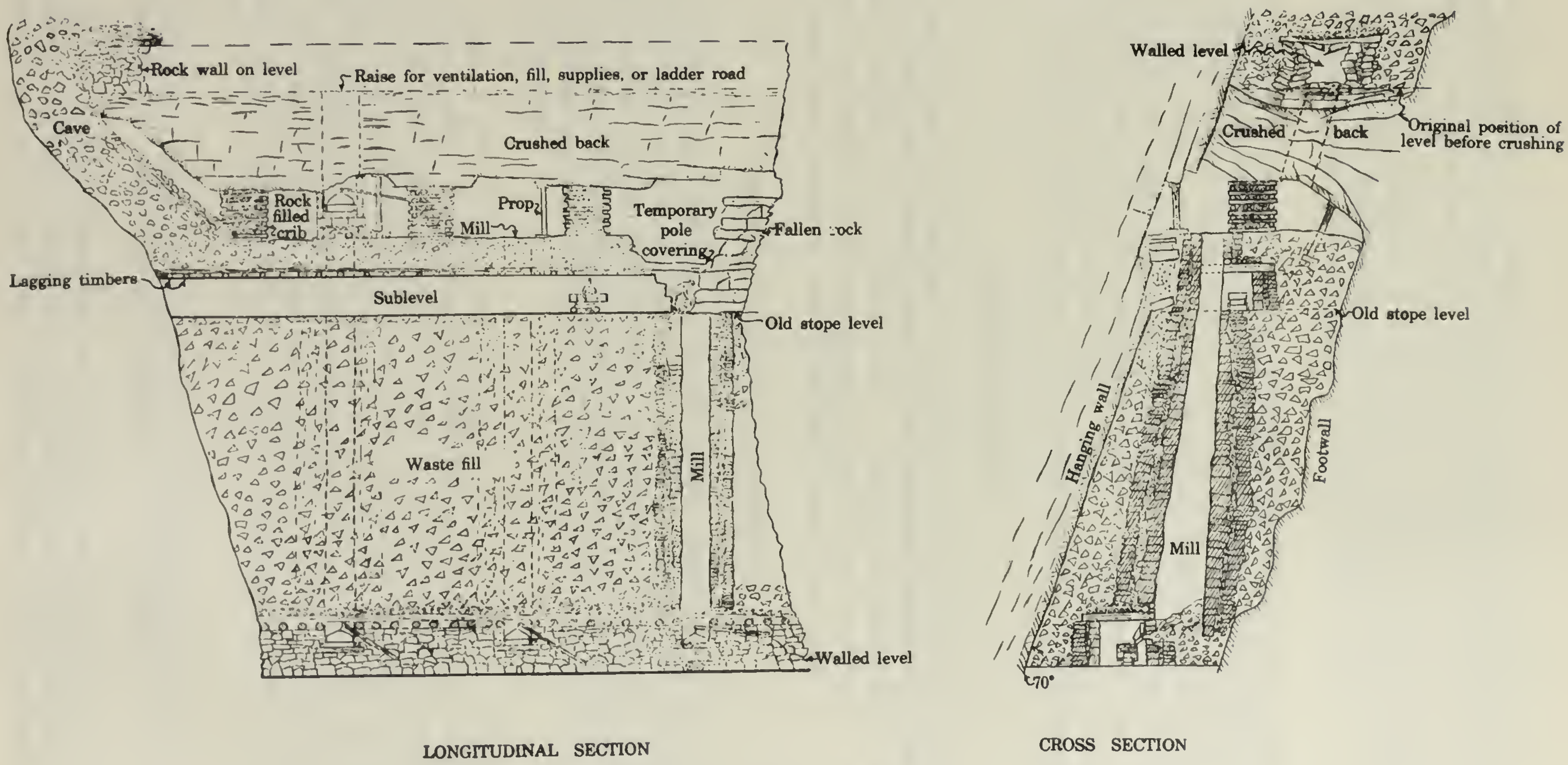


Figure 16.—Recovering crushed floor pillars

It was found that after 55 per cent of the ground in a given area had been mined out the remaining pillars would begin to crush. An attempt to delay this crushing was made at the No. 4 end of Champion mine by allowing the fill to remain packed in the stopes and by dumping fill into the inclined stopes through small raises. A thin floor pillar was left under each level to prevent the level from caving in. By not disturbing the fill the squeeze was delayed a few years, but eventually the ground in this part of the mine started to crush. The method of recovering crushed floor pillars is shown in Figure 16.

The filling used for this method of mining came from barren rock discarded by the pickers in the stopes and waste rock from development openings. The latter was hoisted in the shaft and dumped into a diagonal raise which connected with a raise paralleling the shaft, from which the waste rock could be tapped at each level. For many years, while mining was being done in the upper and richer portions of the lode, there was an insufficient amount of discard and waste rock from development work. To make up this deficiency in fill, stamp sand was brought back from the stamp mill, dumped into the waste raise, tapped off at the levels where needed, and distributed in the stopes by means of compressed air.

New Mining Method

The chief object of changing from mining on the advance to mining on the retreat was the desire to have the crushing occur in the abandoned worked-out areas instead of in those areas where mining is still taking place. Mining on retreat meant but one working place on a level. To increase production from a level it was necessary to adopt a sublevel method, which gives three stopes at the inside end of each level (fig. 17).

Hand Sorting and Selective Mining

There were certain features of the old Baltic mining method which so adequately met the physical conditions encountered in the Champion lode that it was highly desirable not only to retain them in any new method but, if possible, to arrange the stoping so that these features could be used more effectively. Hand-sorting of rock broken in the stopes, introduced in the Champion mine in 1905, solved the difficulty of profitably mining a copper deposit of this character.

The Champion lode is wide, and the copper may occur on the hanging side, on the foot side, or in the middle. It is common to encounter a good stretch of copper on the hanging side of the lode and a good stretch of copper on the foot side of the lode, separated from each other by 4 or 5 feet of barren rock. In some places the copper occurs only on the foot and in other places only on the hanging side. When drifting in the lode in copper ground it is not unusual for the character of the ground to change from good to poor and vice versa several times in the course of a month. Moreover, in the stretches of good copper ground the copper occurs, for the most part, in small nuggets or masses surrounded by areas of barren rock.

When drifting, sorting of the rock is accomplished simply by loading the cars either with poor rock or copper rock and disposing of them as such.

It would be highly undesirable when stoping to send to the surface as copper rock all the rock broken, containing, as it does, over 40 per cent of barren rock. A study of the cost data given at the end of this paper indicates clearly the economics of this problem.

Haulage and hoisting, exclusive of fixed maintenance charges, at present cost 25 cents per ton; preliminary crushing in the shaft house, 6 cents per ton; transportation to the stamp mill, 24 cents per ton; and stamping, 40 cents per ton. This indicates that there is a saving of 95 cents on every ton of barren rock that can be left in the stopes. Furthermore, eliminating barren rock from the rock shipped from the stopes increases its copper content per ton and all subsequent costs, figured on a per pound of copper basis, are decreased proportionately.

Difficulties due to barren rock in the lode are encountered in all mines in this district to a greater or smaller degree. Some of the narrower mines in the district have met the difficulty very successfully by mining around the larger areas of barren rock and leaving these areas as pillars to support the openings, using shrinkage stoping. In the Champion mine, where copper in the foot or hanging is often hidden from the miners by horses of barren rock, it is desirable to give the miners great freedom in the matter of breaking barren rock on either the foot or hanging side of their stopes. This practice has found tens of millions of pounds of copper in this mine. Because the amount of barren rock to be disposed of is increased by this practice and because of the physical conditions in the Champion lode, such as width of lode, dip, and a bad hanging, it was necessary to adopt a filling method. In addition to hand sorting, the sublevel method of mining now practiced, with sublevels separated by 25 feet of rock, permits mining around the larger areas of barren rock.

DEVELOPMENT

Main-level drifts are driven fairly straight and of standard size to the boundary. About 200 feet from the boundary a raise is driven to the level above. Stations are cut in this raise 33 feet and 67 feet from the floor of the main level. From these stations subdrifts are driven in both directions, using 15-hp. slusher engines and scraping directly into the raise. These subdrifts are driven the full width of the copper and explore thoroughly the foot and hanging sides of the lode. When the subdrifts reach the boundary raises are again put up to the level above, and stoping starts from the sublevels at the bottom of these raises. Inclined slices are carried up at an angle of 38°, so that when fill is introduced from above it takes its angle of repose.

STOPING

The subdrift at the bottom of a stope has been cut out to the full width of the copper-bearing part of the lode (see fig. 17). Loose ground is supported on props, with flat 6-inch head boards. The miner drills his round of holes parallel to the angle of repose of the fill, using four, five or six holes across the stope, depending on its width. The burden on these holes is 3 to 4 feet at collar and point. The holes are drilled to a depth of 10 feet. The round is blasted against the fill. The rock broken is picked or sorted, the copper rock thrown into a small car and trammed to the nearest raise, and the barren rock thrown to the foot and hanging of the stope. The miner advances his breast up the slope at an angle of 38° until a small 2 to 4 foot pillar is left, supporting the fill in the worked-out stope above. In wide stopes this small pillar must be left 4 feet thick. The pillar is drilled over, and the holes are left to stand without blasting until all copper rock has been picked out of the stope and all exploration has been done on the foot and hanging sides. The foot sides of some stopes have been mined to widths of over 60 feet before the last pillar was blasted out. When all such work has been completed in a stope the pillar is

blasted, and the fill from above rushes down and completely fills the stope. Most of the copper rock contained in the small pillar last blasted is found at the bottom of the pile of fill. The miner now starts again at the bottom of the slope and goes up with another inclined slice.

The thickness of the burden put on the holes varies; in very lean rock, where a considerable discard is made, a small burden is used to facilitate the work of hand sorting.

Experiments are being made to determine the advisability of using slushers instead of stope cars to transport to the raise the copper rock that has been thrown to the bottom of the stope.

DEEP MINING PROBLEMS

Mining Method

In changing the method of mining consideration was given to those problems that will arise with greater depth. The Champion mine was worked by the Baltic mining method with notable success, and it was only when the difficulties encountered due to crushing ground involved unusual maintenance expenditures that the method of attack was altered. The advancing system of mining was used up to depths of approximately 3,000 feet vertically and about the same depth on the dip. The depth to which the retreating system can be used can be estimated by a study of the Calumet & Hecla Conglomerate mine, where mining is being profitably done on retreat at a vertical depth of over a mile.

At the Champion mine no pillars whatsoever are left, either temporarily or permanently, for purposes of support. About 45 per cent of the area between the longitudinal boundaries of the mine is left as unminable ore. These areas of unminable ore are not made up of small pillars but rather take the form of blocks of large dimensions occurring between the chutes of profitable ground. For example, in one place in the mine, under No. 2 shaft (which has been abandoned), a block 1,500 feet by 1,000 feet after extensive exploration proved to be unminable. In the present workings numerous smaller patches of apparently barren rock are being left as pillars.

Although all the shafts were started in the lode, Nos. 3 and 4 shafts are now about 60 feet in the foot of the lode on account of the flattening which takes place at depth. It is planned to maintain the shafts at least this distance in the foot, and at present there appears to be no reason why the lode can not be mined out over the shaft where the shaft is such a distance in the foot. At No. 1 shaft, from the seventh to the fifteenth levels, the lode was as little as 35 feet to the foot of the shaft and was mined out completely over these eight levels without leaving any shaft pillars. This practice, so far from affecting the shaft unfavorably, rather indicates that all pressure was relieved from the shaft in this area. It seems important not to do such mining, either above or under the shaft, unless there is at least 60 feet of ground between the shaft and the workings.

The question of whether, with funds available for extensive development, it would be advisable to apply the retreating system on the lowest level, first extending stoping from the lowest level upward as well as laterally toward the shaft, can not be applied to this mine, since limits of depth are unknown. It does not seem that such a practice would be advisable in a method such as that adopted by the Champion mine, where filling is used. Our

present method of spreading fill in the stopes is by the use of gravity, and mining from the bottom of the property upward would involve a greater amount of handling of fill. The filling material is supplied by pulling fill from the upper stopes and by broken hanging wall rock.

Although it might seem that the overlying rock pressures would aid in mining the ore, as a matter of fact crushing of the ground creates such a hazard that any benefit from such pressures is more than offset.

The level interval - only 100 feet - would seem to involve an excessive cost of development, maintenance, and equipment, and it is possible that as the mine becomes more extensively developed for mining on retreat this interval could be increased. However, to provide sufficient working places as well as to explore the lode throughly before production starts, it is necessary at present to drive levels not more than 100 feet apart.

Support

Where the shaft was sunk in the lode 100-foot pillars were left on either side. At present No. 1 shaft is over 60 feet to the hanging of the lode and Nos. 3 and 4 shafts over 60 feet to the foot of the lode, and no shaft pillars are left. It appears that had the shafts continued in the lode it would have been necessary to increase the size of the shaft pillars left for support.

Temporary support in the stopes is provided by the filling material; heavy ground in the stopes is caught up on props or cribs until the stope can be filled.

In order that stoping on one sublevel may not interfere with the stoping on the sublevel immediately above the stope above must precede the stope below by about 40 feet. This gives a diagonal line of stope faces, and there seems to be no particular disadvantage in this practice.

The principal lines of weakness in the surrounding rock structure have been determined by experience and recently by a very thorough survey made by Dr. W. R. Crane. These lines of weakness have very definitely affected every operation in our mining method.

Although there seems to be little question but that the support given the hanging wall by the stope filling does not stop the squeeze, it limits the amount of disintegration and caving that can take place in the hanging country. As soon as the fill has been compressed, due to the caving of the hanging, it acts as a support and allows mining to proceed in the stopes before the full weight of the hanging country reaches them. Because of the mineral seams and slips in the hanging wall caving of the hanging trap commences very soon after the lode has been mined.

The sublevel method now employed allows rapid mining of a given block of ground, and rapid stoping is a decided advantage in avoiding the development of rock pressure and crushing.

The small pillars of barren rock left in the stoping operations seem to crush later without causing any disturbance in the working faces. The barren blocks of large dimensions, some of which cover many thousand square yards in area, carry the weight of the hanging

country for a considerable time, and when one of these crushes there follow what is commonly known as an "air-blast" and extensive shaking up of the ground in its neighborhood.

Hoisting and Pumping

There is very little water in the lower levels of the Champion mine, and the problem of pumping at depth is not yet serious.

It appears that with depth and the possibility of crushing reaching the shaft, causing uneveness in the road bed and delays for repairs, hoisting speeds will be reduced rather than increased and greater emphasis placed on sorting and selection of rock before it is hoisted.

UNDERGROUND HAULAGE

On the upper levels of the mine the track gage is 24 inches. Two-and-one-half-ton cars are used for tramming copper rock to the shaft and for hauling filling material from the waste raise. The chutes used in the older parts of the mine are shown in Figure 15.

The copper rock is dumped into the skip by dumpers, two cars to a skip, by means of a turntable and cradle. Waste rock for filling is dumped down into the raises between the rails, the cars being up-ended by 10-inch air lifts.

The mine is equipped with seventeen 4-ton, 15-hp. General Electric locomotives, twenty-four 4-ton, 30-hp. Goodman locomotives, and one Jeffrey locomotive. In the lower levels the track gage is 36 inches.

The track, trolley, and all haulage equipment are put in and maintained according to printed standards. Strict attention is given on all levels and in all stopes to "good housekeeping."

The 5-ton cars used on the lower levels and the arrangements for dumping them are shown in Figure 7. The timbered chutes at the bottoms of the raises are shown in Figure 14.

PERCENTAGE OF EXTRACTION

Of the rock broken in the mine for all operations in 1930, 56 per cent was sent to surface as stamp rock; the balance was left in the mine as stope filling. At present very little waste rock is dumped on surface, but the amount is increasing.

WAGE CONTRACT AND BONUS SYSTEM

With the exception of repair and maintenance gangs all men in the mine work on contract. About 71 per cent of the men on the underground payroll make money on their contracts; the average pay per contractor is about 50 cents per shift above the "company-account" base.

Shaft Sinking

Shaft sinking is paid for at a set price per foot sunk and covers mining, mucking, and timbering labor. Up to a limited number of pounds per foot sunk powder is free; all powder used above that limit is paid for at cost by the contractors.

Drifting (Main-Level and Sub Drifts

Drifting is paid for at a set price per foot of advance, the price varying for each different width of drift and covering labor of mining, mucking and disposing of rock. Up to a limited number of pounds per foot of advance powder is free, the amount allowed varying with each different width of drift; any excess powder is paid for at cost by the contractors.

Stoping

Because hand sorting is practiced in the mine the stoping contracts are somewhat more complicated than the development contracts. Barren rock must be kept out of the copper cars, and copper rock must not be discarded as waste. The rule given to pickers is simple: Large pieces of rock containing any copper whatsoever that can be seen must be shipped as copper rock; fine rock must be shipped as copper rock unless ordered left in the stope by the boss. Cars of copper rock are inspected by the bosses and by a poor-rock inspector. Discard in the stopes is inspected by the bosses and the captain.

In the stopes the contract system is balanced in such a way that the best pay will be made by those contractors who carefully sort their rock. The organization in a stope consists of a miner and one or two pickers on each shift. The miners' company-account rate is 45 cents a day above the pickers' company-account rate. These men mine, pick, and do all the propping necessary in their stope. Each participates in the bonus earned for the month in proportion to the number of shifts he has worked. Engineers measure monthly the volume of the excavation made in each stope to determine the total rock broken in each stope and the average width of the stope. Underground records show the number of cars of copper rock shipped from each stope. The time books show the number of shifts of all labor working in each stope. From these data the engineers calculate monthly the bonus to be paid to each party of contractors.

The use of three tabulated rate schedules facilitates the work of calculation. The rate schedules cover:

1. The bonus per shift to be paid for a given tonnage of rock broken in a stope of a given width; the wider the stope, the greater the tonnage broken per man per shift for a given bonus.

2. The bonus per shift for a given tonnage of copper rock produced, after sorting, from a stope showing a given ratio of total broken to total copper rock produced; the greater the amount of barren rock that must be handled in sorting, the greater the bonus pay per shift for a given tonnage of copper rock produced.

3. The allowance of powder per ton of rock broken for each width of stope.

VENTILATION

The subject of ventilation in the Champion mine has recently been thoroughly covered by a United States Bureau of Mines information circular now in preparation, Natural Ventilation in the Michigan Copper Mines, by G. E. McElroy.

The Champion mine is comparatively shallow. So far good ventilation has been maintained in all working places; even the blasting may be done at any time during the working shift. Analyses of the air in what were considered the worst stopes and drifts showed no carbon monoxide and only a very slight deficiency in oxygen.

FIRE HAZARDS

The greatest fire hazards in the mine are faulty electrical installations and open lights. Smoking underground is strictly prohibited.

A well-equipped mine rescue-station is maintained, containing six McCaa 2-hour oxygen apparatus and several all-service gas masks. Twenty-four picked men have received United States Bureau of Mines certificates in mine rescue-training and are given additional training each month by the safety inspector. Warning to the men in the mine in case of fire has been provided for by making suitable connections in the air main for the introduction of stench. In the mine, water barrels and buckets, plainly marked "FIRE", are placed on each plat, directly opposite the main hoisting compartment. These barrels are maintained in perfect condition and serve, not only as a protection in case of fire, but also as a constant reminder to the men. Forty-three 1½-quart, Pyrene extinguishers and six Foamine, 2½-gallon extinguishers are placed at hazardous points in the mine. In Nos. 1 and 4 shafts, water lines with suitable fire-hose connections are run from the surface to the bottom of the mine. Small tanks with overflow connections are placed on every third level to break the pressure in these water lines. Bulkheads and doors have been constructed to cut off certain sections of the mine in case of fire.

SAFETY METHODS AND FIRST AID ORGANIZATION AND TRAINING

Accident-prevention work at the Champion mine is regarded as a major operation by the president and all company officials. The safety organization consists of a general safety committee, of which all department heads are members, having direct charge of accident prevention work. In addition, the mine employs a full-time safety inspector.

The results of the aggressive safety campaign which was started in 1927 are indicated in the following table, giving the accident record since 1926:

Year	Total of all lost-time accidents
1926	430
1927	339
1928	254
1929	172
1930	92
1931 (3 months)	11

The accident-prevention work takes the form of repeated inspections, investigations of all accidents and near accidents, mimeographed safety news letters every two weeks to all foremen, well-conducted bulletin-board service, rigid enforcement of safety rules, and periodical examinations of the men to determine their understanding of the safety rules, regular

safety meetings for foremen, and individual awards to foremen and men who maintain good no-accident records. Hard-toed shoes, hard hats, and goggles must be worn by all men in the mine. Gloves must be worn by men doing certain kinds of work.

All the men in the mine have received certificates of training in first aid from the United States Bureau of Mines. This training was done in 1930, and since then all new men who have been employed have been trained and received certificates. For the purpose of reviewing first-aid training classes are held periodically, and all men are requested to attend. All first-aid training is taken on the men's own time. The same training was given to all men employed in the surface plants. First-aid boxes containing the necessary bandages, etc., are stationed on every level underground and at suitable places on the surface.

Table 1.- Summary of Costs at Champion Mine

Period: Five months—August 1, 1930, to December 31, 1930.

Ore hoisted during period:---193,597 tons.

Mining method: Inclined cut-and-fill, partly from sublevels, retreating.

Underground Costs Per Ton of Ore Hoisted

	Labor	Supervision	Compressed air, drills, and steel	Power cost	Explosives	Timber	Other supplies	Total
Development	$0.297	$0.023	$0.069		$0.069	$0.031		$0.490
Mining (stoping)	.794	.056	.128		.103	.061	$0.020	1.163
Transportation underground & hoisting	.390	.027		$0.089		.017	.029	.552
General underground expense	.045			.008			.021	.078
Mine unwatering	.027	.001		.095			.010	.133
Surface expense directly applicable to underground operations	.091						.021	.112
Total	1.644	.107	.197	.192	.172	.109	.101	2.528
Cost per pound of copper	.036	.002	.004	.004	.003	.002	.002	.055

Note:- About 40 per cent of rock broken is sorted out underground as waste.

Table 2.- Summary of Costs in Units of Labor, Power, and Supplies

Champion mine August 1, 1930, to December 31, 1930.

Tons of copper rock hoisted: 193,597 tons of 46 pounds copper each.

Mining method: Inclined cut and fill, partly from sublevels, on retreat.

A.-Labor (man-hours per ton)	Development	Mining	Total
Breaking (drill and blast)	0.18	0.55	0.73
Timbering and filling	.09	.21	.30
Shovelling and hand sorting, underground	.18	.73	.92
Haulage and hoisting	.04	.82	.86
Supervision	.03	.09	.12
General	.02	.10	.11
Total labor underground	.54	2.50	3.04
Average tons per man per shift	14.6	3.17	2.64
Labor —— per cent of total cost	62	67½	66
Average tons per man per shift on surface charged underground			65
B.-Power and supplies, per ton			
Explosives, per ton of ore ... lbs.	.5	.8	1.3
Timber (board measure), per ton of ore ... ft.	1.2	3.1	4.3
Total power ... kw.h.per ton			21.7
Air compression	4.0	8.4	12.4
Hoisting	.7	3.3	4.0
Pumping			4.7
Ventilation, lighting, etc.			1.0
Other supplies —— per cent of total supplies and power			48.5
Supplies and power —— per cent of total cost			17.8
C.-Percentage of total cost			83.8

Table 3.- Detail of Costs in Units of Labor, Power, and Supplies

	Sinking	Drifting	Crosscuts	Raising	Total all development
Size of excavation.................. feet	10 by 23	8 by 13½	-	4 by 8	-
Timbered or not..................	Timbered	Part	-	Timbered	-
Physical properties of rock..................	Hard, loose	Hard, firm	Hard, firm	Hard, firm	
A.-Labor--Man-hours per ft.					
Breaking (drilling and blasting)..................	14.7	4.9	-	6.6	5.8
Timbering..................	36.3	1.2	-	-	2.8
Shovelling..................	23.1	6.2	-	-	5.9
Haulage and hoisting..................	16.0	.8	-	-	1.4
Supervision..................	1.1	1.1	-	1.1	1.1
Total labor..................	91.2	14.2	-	7.7	17.0
Feet per 8-hour shift..................	.088	.56	-	1.0	.47
B.-Power & Supplies (per ft.)					
Explosives..................lbs. per ft.	25	17.0	-	14.	17
Timber, (b.m.) per ft.................. ft.	330	11	-	8	28
Total power..................kw.h.per ft.					
Air compression..................	-	-	-	-	130
Hoisting..................	-	-	-	-	27
Haulage and ventilation..................	-	-	-	-	4.5
Other supplies..................					
Labor --- per cent of total cost..................	-	-	-	-	60.5
Power and supplies --- per cent of total cost..................	-	-	-	-	39.5

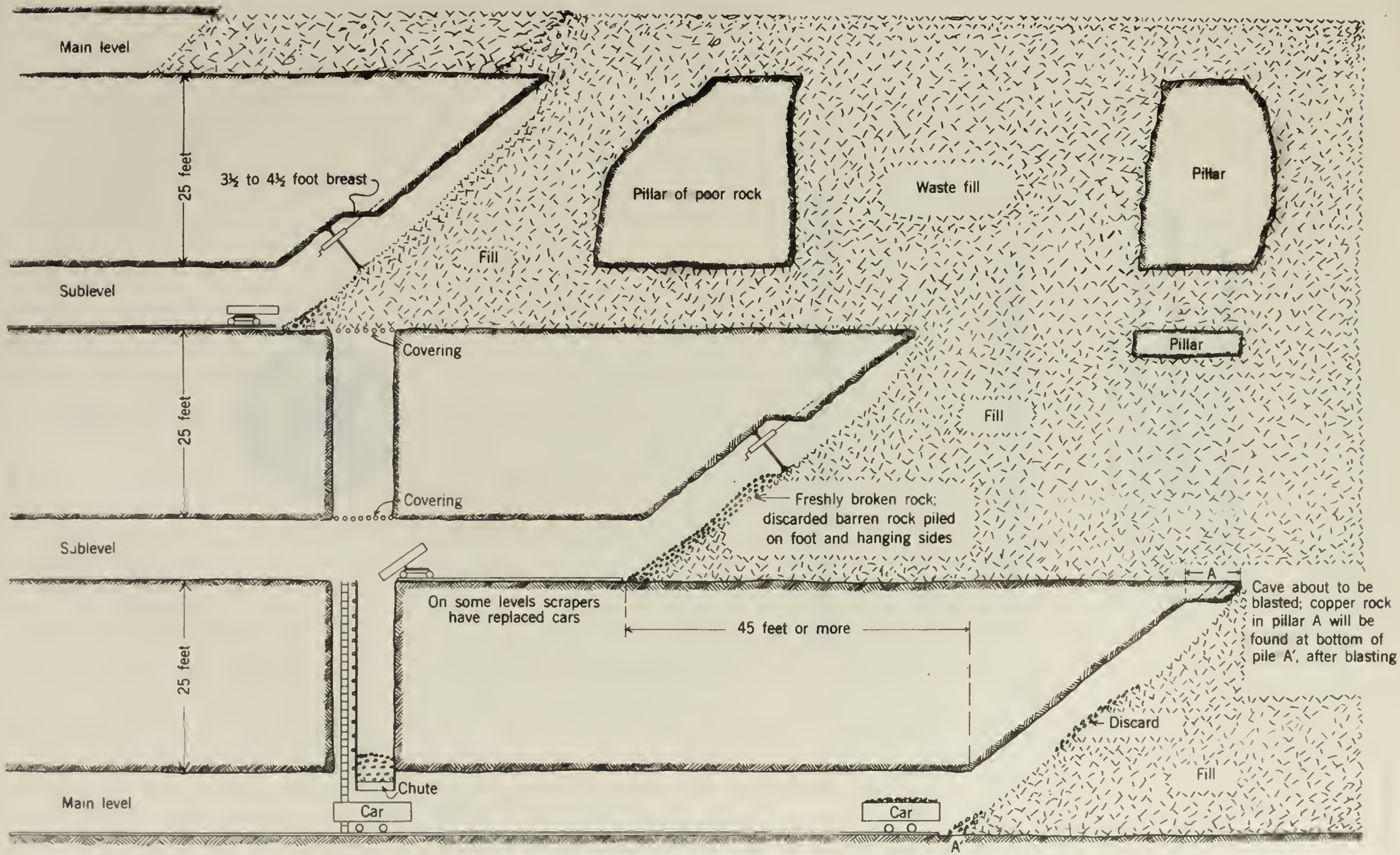


Figure 17.—New stoping method—sublevel, inclined cut-and-fill

I. C. 6516

October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF GREAT BRITAIN

BY

E. P. Youngman

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

MINING LAWS OF GREAT BRITAIN[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining lesigla-tion and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of England was guided by a statement and memorandum formulated in the office of the Mines Department, London, and forwarded through the American Embassy, in reply to a questionnaire submitted by the United States Bureau of Mines and transmitted through the courtesy of the State Department.

INTRODUCTION

Great Britain has never had a basic mining law such as is in force in most foreign countries. This lack may be due to the fact that, with the exception of gold and silver, all metals or minerals belong to the owner of the surface land. Even with respect to the precious metals, gold and silver, which are not abundant in Great Britain, the right of the Crown, though still technically acknowledged, is not always insisted upon. All other metals and minerals are vested in the ground landlord, who may sell, lease, or other-wise convey them.

The State has little to do with the mining industry except in regulatory and judicial matters. The Government must settle controversies and provide for easements, rights of way for railroads, roads, and canals, or, in general, effect the same control that it exercises over other industries.

As mining was carried on in very early times in England, of necessity a certain legal procedure came into existence, which in effect was little more than a private contract between the owner of the mineral (whether an individual or the State) and the miner. The nature of such contracts, of course, has changed radically with the changed relations between classes, as England has developed from the feudal into the modern State. From the medley of ancient procedure and custom, a concession system of mining in its simplest form has evolved. Of the earlier conditions, Van Wagenen says:[3]

1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6516."
2 - Rare metals and nonmetals division.
3 - Van Wagenen, Theo. F., International Mining Law: McGraw-Hill Book Co. (Inc.), 1918, pp. 221-223.

The tin mines of Cornwall and Devonshire, the coal and iron mines of Gloucestershire, and the lead mines of Derbyshire are operated to-day under very ancient customs, not very different from those that were in force in Germany and other parts of the continent during the twelfth and thirteenth centuries. In Cornwall and Devon an institution called the Stannary Court (abolished in 1896, jurisdiction being given to county courts), the origin of which is lost in antiquity, exercised a limited degree of control between surface owners and lessees, the principal effect of which was to compel the former either to work the mines themselves or to allow others to do so upon the payment of a reasonable rent or royalty. In Derbyshire there still exists a somewhat similar organization, by the acts and decisions of which the very ancient right of following the veins of the region on their dip outside of surface side lines, in accordance with the principles in force in central Europe during and before the fifteenth and sixteenth centuries, has been preserved to the locality.

A publication of 1930 (Complete Statutes of England), referred to in the following section of this paper, fully discusses the ancient customs still in force in Cornwall (with respect to tin mines), in Derbyshire (with respect to lead mines), and in Gloucestershire (principally with respect to iron mines).

SOURCES OF INFORMATION

No legislation with special reference to the mining industry in England was passed until 1850; the law then enacted had to do with contractual relations between owners and lessees and furnished means of arbitration in the case of disputes. A list of legislative acts bearing upon mining from 1688 to 1930, which follows, speaks for itself with respect to the subjects legislated upon. Countless statutory laws dealing with other subjects (wages, accidents, employment of children, weights and measures, and so on) are applicable to the mining industry; and the decisions of numberless case laws are effective. The most recent and probably the most important of the mining acts are the mines (work facilities and support) act of 1923; the mining industry act of 1926; and the coal mines act of 1930.

For a comprehensive study of the collective mining laws of Great Britain, the reader is referred to the sections entitled "Mines, Minerals, and Quarries" in Halsbury's Laws of England[4] and the Complete Statutes of England.[5]

A book published by Watson,[6] which covers the various acts of Parliament with respect to the regulation of mines, contains the following table of contents.

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4 - Halsbury, Earl of (Stanley Giffard Hardinge, 1st), Halsbury's Laws of England: Vol. 20, London, 1911, pp. 497-650.

5 - Chitty, Sir Thomas Willis, and others, Complete Statutes of England (Continuation of Halsbury's Laws of England): Vol. 12, London, 1930, pp. 3-208.

6 - Watson, Alexander, Mining Law and Mine Management; A Textbook for Candidates for the Manager's and Under-Manager's Certificate: Sir Isaac Pitman & Sons (Ltd.), Toronto, or New York, 1927, 223 pp.

Mines officials and the law.
Firemen, examiners, or deputies.
Enginemen and workmen.
Safety provisions.
Provisions as to shafts and outlets.
Miscellaneous safety provisions.
Health, accident, and rescue provisions.
Inspectors, prosecutions, etc.
Acts relating to weighing of minerals.
Coal mines regulations act, 1908 (with amendments).
Coal mines (minimum wage) act, 1912 and 1920.
General management of collieries.
Distribution of labor.
Organization for production.
General management.

Mining Laws of Great Britain

1688	1W&M.ch.30.sec.4.
1693	5W&M.ch.6.
1799-1800	(E.)39-40G.3.ch.77.sec. 4,6,9,10. (Offences.)
1814-1815	(E.)55G.3.ch.134.
1836	(E.)6-7W.4.ch.19.
1855	Stannaries act, 1855. 18-19V.ch.32.
1857-1858	(E.) 21-22V.ch.45.sec.5.
1861	(E.N.I.) 24-25V.ch.96.sec.39. (Offences.)
1861	(E.N.I.) 24-25V.ch.97.sec.26-29 (Offences.)
1866	29-30V.ch.62.sec.21-25. (Foreshore.)
1869	Stannaries act, 1869. 32-33V.ch.19.
1872	Metalliferous mines regulation act, 1872. 35-36V.ch.77.
1875	38-39V.ch.17.sec.59. (Explosives.)
1875	Metalliferous mines regulation act, 1875. 38-39V.ch.39.
1878	41-42V.ch.49.sec.65,86. (Weights and measures.)
1882	Slate mines gunpowder act, 1882. 45-46V.ch.3.
1884	(E.) 47-48V.ch.43.sec.4. (Summary proceedings.)
1887	Coal mines regulation (check weighers) act, 1887. 50-51V.ch.58. sec.1,3,12-15.
1887	Quarry fencing act, 1887. 50-51V.ch.19.
1887	Stannaries act, 1887. 50-51V.ch.43.
1891	Metalliferous mines (Isle of Man) act, 1891. 54-55V.ch.47.
1891	Brine pumping (compensation for subsidence) act, 1891. 54-55V.ch.40.
1894	Quarries act, 1894. 57-58V.ch.42.
1894	Coal mines (check weighers) act, 1894. 57-58V.ch.52.
1900	Metalliferous mines (prohibition of child labor underground) act, 1900. 63-64V.ch.21.
1905	Coal mines (weighing of minerals) act, 1905. 5E7.ch.9.
1906	Metalliferous mines (notice of accidents) act, 1906. 6E7.ch.53,sec.1,3,5.
1908	Coal mines regulation (hours below ground) act, 1908. 8E7.ch.57.
1910	Metalliferous mines (accidents, rescue, and aid) act, 1910. 10E7.&1G5.ch.15.

1911	Coal mines (consolidation) act, 1911. 1-2G5.ch.50.
1912	Coal mines (minimum wages) act, 1912. 2-3G5.ch.2.
1914	Coal mines act (amendment of 1911 act), 1914. 4-5G5.ch.22.
1916	Coal mines (certificates of competency) act, 1916. 6-7G5. ch.31.sec.10.
1916	(E.N.I.) 6-7G5.ch.50.sec.11. (Larceny.)
1916	Police, factories, etc. (miscellaneous provisions) act, 1916. 6-7G5.ch.31.
1917	Coal mines regulation (hours below ground) act, 1917. 7-8G5.ch.8.
1918	(E.) 8-9G5.ch.39.sec.14. (Education.)
1918	(S.) 8-9G5.ch.48.sec.17. (Education.)
1918	Petroleum (production) act, 1918. 8-9G5.ch.52.
1919	Coal mining (hours below ground) act, 1919. (E.S.) 9-10G5.ch.48.
1919	Checkweighing in various industries act, 1919. 9-10G5.ch.51.
1920	Coal mines (emergency) act, 1920. (Control and profits.) 10-11G5.ch.4.sec.7-10.
1920	Mining industry act, 1920. 10-11G5.ch.50.
1920	10-11G5.ch.65.sec.1,3-5. (Employment of women, young persons, and children.)
1921	Coal mines (decontrol) act, 1921. 11-12G5.ch.6.
1923	Mines (working facilities and support) act, 1923. 13-14G5.ch.20.
1923	13-14G5.ch.42.sec.28. (Accidents.)
1925	Mining industry (welfare fund) act. 15.16G5.ch.80.
1925	15-16G5.ch.84.sec.15,34. (Workmen's compensation.)
1925	Mines (working facilities and support) act, 1925. (Amendment to 1923 act.) 15-16G5.ch.91.
1926	Coal mines act, 1926. (Temporary amendment to coal mines acts, 1887-1917, with respect to hours of employment below ground.) 16-17G5.ch.17.
1926	Mining industry act, 1926. 16-17G5.ch.28.
1930	Coal mines (reorganization of industry) act. 20-21G5.ch.34.

CLASSIFICATION OF MINERALS

There is no definite classification of minerals with respect to the mining laws. There is a certain separation, however; so that the legislation falls into the following divisions: metalliferous mines, coal mines, petroleum mines, and quarries.

The statutory law regulating metalliferous mines is embodied chiefly in the metalliferous mines regulation acts of 1872 and 1875, which apply to every mine of whatever description other than a mine to which the coal mines regulation act of 1887 applies. In many respects the statutory provisions are identical with the coal regulations.

The statutory law regulating coal mines and mines of stratified iron-stone and mines of shale and mines of fire clay is embodied in the coal mines acts of 1887 to 1930, which, with regard to mines in the Forest of Dean,

are supplemented by special acts confirming customs existing among the people and miners of the forest from time immemorial.

The petroleum production act of 1918, governs certain phases of oil production.

The regulation of quarries (including every bed or opening made for the purpose of getting stone, slate, lime, chalk, clay, gravel, or sand) is provided for chiefly by the quarries act of 1894, which makes applicable to them certain provisos of the metalliferous mines regulation acts of 1872 and 1875.

Lists of acts dealing with the four classes of mines follow.

Metalliferous Mines Acts

Metalliferous mines regulation act, 1872.
Metalliferous mines regulation act, 1875.
Metalliferous mines (Isle of Man) act, 1891.
Metalliferous mines (prohibition of child labor underground) act, 1900.
Metalliferous mines (notice of accidents) act, 1906.
Metalliferous mines (accidents, rescue, and aid) act, 1910.

Coal Mines Acts

Coal mines regulation act, 1887.
Coal mines (check weigher) act, 1894.
Coal mines (weighing of minerals) act, 1905.
Coal mines regulation act, 1908.
Coal mines regulation act (suspension in emergency), 1908.
Coal mines act, 1911.
Coal mines act (application of), 1911.
Coal mines (minimum wage) act, 1912.
Coal mines act, 1914.
Coal mines regulation act (amendment), 1917.
Coal mines act, 1919.
Coal mines (emergency) act, 1920.
Coal mines (emergency) act (curtailment of), 1920.
Coal mines (decontrol) act, 1921.
Coal mines act, 1926 (amendment of acts 1887-1917, with respect to hours of employment underground).
Coal mines (reorganization of industry) act, 1930.

Petroleum Act

Petroleum (production) act, 1918.

Quarries Acts

Slate mines gunpowder act, 1882.
Quarry fencing act, 1887.
Quarries act, 1894.

MINING AUTHORITY

The central authority in mining matters is the Mines Department (of the Board of Trade), which is represented in the cabinet by an undersecretary of mines. (Pt. 1, mining industry act, 1920.)

RIGHTS OF FOREIGNERS

No discrimination against Americans or other foreigners exists in the mining laws of Great Britain. Foreigners are permitted to negotiate, on equal terms with British subjects, with the owners of minerals for the right to prospect or to mine. The terms of a prospecting or mining lease are determined by those interested.

With respect to Crown-owned minerals, gold and silver and all minerals under soil that is owned by the Crown, the right to explore or mine is obtained by negotiation with the Government department administering the minerals in question.

In the case of coal, any one taking an active part in the management of a coal mine must hold a certificate of competency, which involves the passing of an examination in English mining legislation.

Incorporation under the laws of England is not required.

PROPERTY IN MINES

The ownership of the surface and that of the subsoil may be distinct; "and the mines so severed are a separate tenement, capable of being held for the same estates as other hereditaments and with the like incidental rights of ownership." A separation may be effected by conveyance or by an act of Parliament. In the absence of evidence the severance may be inferred from long and continuous possession of the mines by persons other than the owners of the surface. Although mines may be thus obtained, yet, prima facie, an owner of the surface is entitled, ex jure naturae, to all beneath it, except mines of gold and silver; and it is immaterial that a title has been acquired only by acts of ownership on the surface. If an estate in fee simple in land is acquired by enlargement of a long term, the estate thus acquired includes the mines if they are not separated at the time the enlargement is made. But proof of ownership of the mines under any parcel of land does not raise any presumption or afford any evidence requiring ownership of the surface.

At common law, the Crown, by virtue of royal perogative, was entitled to all mines containing gold and silver. This position was modified by statute (1688); so the Crown now claims only those mines that are worked primarily for gold and silver; where the baser metals predominate, payment must be made.

Mines under navigable rivers, under the shore between low-water mark and the ordinary high-water mark (known as foreshore), and under the bed of the sea adjoining the shore belong, prima facie, to the Crown. Mines under non-navigable rivers belong to the owners of the adjoining lands. Mines under the foreshore may, however, be shown to belong to the adjoining owner.

PROSPECTING AND MINING

In a country like Great Britain, where the mineral deposits are, generally speaking, already known, the question of prospecting as distinct from such measures as trial borings does not really arise. Moreover, prospecting rights are the result of agreements between the owner of the surface and the prospective explorer, as are also mining rights, in the case of private property, and the result of negotiations with the department administering Crown-owned minerals, in the case of State property. Consequently, compensation for ground occupied by mining works or damaged during mining operations is assured to the ground owner.

Alford,[7] writing in 1906, gives an example of the "takenote" (pronounced "tacknote") that has been used in Cornwall and in other parts of England. The takenote, Alford says, carries with it far larger powers of working and disposal of minerals than do the modern prospector's licenses of the British colonies and other countries and is in itself a kind of mining lease under which real mining and commercial operations can be carried on--the only great difference being that the lease is for a much-extended period of time and that it is more elaborately drafted. "In any case, the takenote secures to the grantee the right to a mining lease whenever it becomes desirable to obtain one."

Of the mining leases of Great Britain, of which he gives an example, Alford[8] says:

> The mining leases of Great Britain are frequently very wonderful and elaborate documents, engrossed on immense sheets of parchment and often encumbered with complicated clauses not only inapplicable to the usages of mining but also most pernicious to the interests of both parties. In many cases the leases are drawn by the solicitors of the lord without any reference to the grantee or to any person acting for him, with the result that the greater part of the lease remains forever a dead letter, and the work is carried on under it in the thoroughly English give-and-take fashion, by personal, and very frequently only verbal, agreement between the lord and his agent and the mining company's manager. This peculiar system appears to work well enough in England between Englishmen. . . .

Leases

The minerals that are to be included in an agreement for a lease must be properly defined; otherwise, the agreement will not be binding.

An agreement for a lease shall contain stipulations as to the rents and royalties to be reserved by, as well as the covenants and provisions to

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7 - Alford, Charles J., Mining Law of the British Empire: Charles Griffin & Co. (Ltd.), Exeter Street, Strand, London, 1906, pp. 12-14.
8 - Alford, Charles J., Work cited, pp. 14-23.

be inserted in, a lease. Details concerning rights included in and those implied by a lease may be found in Halsbury's Laws of England.[9]

Licenses[10]

A right to work mines and appropriate the minerals won is more than a license. "It is a profit à prendere lying-in grant," which may be limited by either freehold or chattel interests." Estates thus created may be devised, inherited, or assigned. "It does not convey any estate in the land or in the mines except the parts severed," which become the property of the grantee. Such a license is irrevocable. A licensee that has entered into possession is liable in an action for use and occupation; and he may if ousted, at any rate where the license is exclusive, bring an action to recover possession.

A bare license (such as a license to search for minerals), which does not amount to a profit à prendere, confers no property rights in the minerals uncovered. A grantee has the right of possession for the purpose of examining the minerals found and has the right of action against an interfering third person. A bare license (granted by deed or not, for valuable consideration or not) is revocable. However, revocation of the license may entitle the licensee to damages for breach of contract.

A mining license, coupled with a grant, must be reacted by deed, in order to create a legal right; and it can be legally transferred by deed.

Licenses that amount to a profit à prendere are usually made in a form similar to mining leases and contain similar covenants and provisions--covenants, for instance, to pay for surface damage. It is usual to insert a proviso for reentry.

An exclusive license is one that gives to the grantee a sold right within defined limits. If this sole right is not clearly expressed, the grantor may exercise the same right and authorize others to do so, provided that neither he nor they interfere with the licensee or deprice him of the benefits of his license. "An exclusive license may maintain trespass."

DAMAGES

Compensation for any damage caused by reason or in consequence of the working and getting of the minerals to lands, hereditaments, buildings, works, or structures is payable by the applicants, and any differences arising therein is referred to the commission (Railway and Canal Commission).

Where a person to whom any compensation or consideration is payable can not be found or ascertained, the amount due is paid into the Court of the Railway and Canal Commission.[11]

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9 - Halsbury, Earl of (Stanley Giffard Hardinge), Work cited, pp. 556-567.
10 - Halsbury, Earl of, Work cited, pp. 567-570.
11 - Bamber, Philip Gordon, The Mines (Working Facilities and Support) Act, 1923 (with an appendix on the practice of the Railway and Canal Commission Court): The Colliery Guardian Co. (Ltd.), 30 Furnival Street, Holborn, E. C. 4, London, 1925, pp. 66-68.

TRANSFERS

The rules of law applicable to contracts for sale or demise of land apply generally to contracts concerning mines, quarries, and unsevered minerals, if the mines and minerals are part and parcel of the land.

FORFEITURE

It is now usual to insert in mining leases a power of reentry, enforceable not only upon nonpayment of rent but also upon breach of covenant or condition. Sometimes the power is made applicable also if the mine is not worked for a specific period, if the lessee becomes bankrupt or makes an arrangement with his creditors, or if the lessee (being a company) goes into liquidation. Sometimes the lease contains its provisions for the voidance of the lease.

REORGANIZATION OF THE COAL INDUSTRY

The production of coal in Great Britain is now regulated by the coal mines act of 1930, which (in part 1) established a Central Council (composed of representatives of the colliery owners throughout the country), which decides periodically what amount of coal shall be produced by the industry as a whole. The council divides the total quantity of coal to be produced among the various districts, in each of which an executive board represents the colliery owners.

The duty of each executive board is to divide fairly among the colliery owners the allocation made to its district, according to the methods prescribed by the scheme administered by the board (no scheme to have effect unless approved or made by the Board of Trade), and to determine minimum prices for each class of coal. Appeal by any one executive board may be made against the Central Council, against another executive board, or against any individual to independent arbitrators provided for in the scheme in force in the particular district.

Committees of investigation have been provided for, in accordance with the act, to look into complaints made against the schemes in force. A national committee shall be charged with the duty of investigating any complaint made with respect to the operation of the central scheme. A district committee, for each district, shall be charged with the investigation of complaints made against the operation of a district scheme. No committee shall investigate a complaint that could be referred to arbitration. The chairman and the members of each committee shall be appointed by the Board of Trade. Of the members of the committee, other than the chairman, one-half shall be persons appointed to represent the interests of the consumers of coal. In the case of the national committee (consisting of nine members), two persons shall represent owners of coal mines in Great Britain, and two shall represent workers employed in or about the coal mines. In the case of a district committee (consisting of five members) one person shall be appointed to represent the owners of coal mines in the district and one to represent workers employed in or about the coal mines. The investigating committees report to the Board of Trade, which is charged with making recommendations to the

executive board concerned and, in the event that its recommendations are not carried out, with annulling the scheme in effect and subsituting a new one. Any person aggrieved by the neglect of an investigating committee or by any act or omission of the Board of Trade may appeal to the Railway and Canal Commission.

Part 2 of the coal mines act of 1930 provides for the appointment by the Board of Trade of a commission of five members, to be known as the Coal Mines Reorganization Commission, the duty of which, in general, seems to be the reorganization of the coal-mining industry with a view to facilitating the production, supply, and sale of coal by owners of coal mines, and, in particular, to promote and assist, by the preparation of schemes and otherwise, the amalgamation of undertakings of or comprising coal mines where such amalgamations appear to the Commission to be in the national interest. (Part 2 of the act of 1930 is an amendment to the mining industry act of 1926.)

The mining industry act of 1926 (with respect to means of facilitating the working of minerals, the better organization of the coal-mining industry, and the welfare of persons employed therein) enables one or more colliery owners to obtain the compulsory absorption of neighboring collieries. Application is made to the Board of Trade by the submitting of an absorption scheme (total amalgamation scheme, total absorption scheme, or partial amalgamation or absorption scheme). The Board of Trade, if convinced that a prima facie case has been made out, refers the application to the Railway and Canal Commission, which, after hearing all objectors, may confirm the scheme, with or without modification, or refuse it. Whereas the act of 1926 is in the interests of the colliery owners desiring to effect an amalgamation, the act of 1930 is in the national interest; for the Coal Mines Reorganization Commission, under that act, may instigate amalgamation schemes "if it appears that it is expedient for the purpose of promoting the more economical and efficient working, treating, or disposing of coal that an amalgamation scheme or an absorption scheme (under part 1 of the mining industry act, 1926) should be prepared. . . ." If at the suggestion of the Reorganization Commission the owners are not willing to amalgamate, the commission is authorized to prepare and submit its own scheme to the Board of Trade, which submits it to the Railway and Canal Commission. If the owners, of their own accord, submit a scheme that the Reorganization Commission certifies to be in the national interest, it need not be referred to the Railway and Canal Commission, but only to the Board of Trade.

Part 4 of the coal mines act of 1930 provides for the appointment (by the Board of Trade) of a Coal Mines National Board, of 17 members. Prior to the appointment of the members (other than the chairman) of this board, the Mining Association of Great Britain shall be consulted as to 6 of the members, the Miners' Federation as to 6, the Federation of British Industries and the Association of British Chambers of Commerce as to 1, the General Council of the Trade Union Congress as to 1, the Cooperative Union as to 1; and the National Confederation of Employers' Organizations as to 1. Agreements, between owners of coal mines and the workers, providing for the regulation of wages or other conditions of labor, may be sent to the National Board, with which such agreements must be recorded. Either the owners or the workers may refer the dispute.

PETROLEUM

No person other than a person acting on behalf of His Majesty, or holding a license under the petroleum act of 1918 (8 and 9 Geo. 5, ch. 52) for the purpose, shall search or bore for or get petroleum within the United Kingdom; and if any person gets petroleum in the United Kingdom in contravention of this provision, he shall forfeit to His Majesty a sum equal to three times the value of any petroleum obtained by him.

The Minister of Munitions, on behalf of his Majesty, may grant licenses conferring authority to search and bore for and get petroleum to such persons and upon such terms as the Minister of Munitions may think fit. When any such license is granted, a copy thereof shall be laid before Parliament as soon as may be after the grant thereof.

For the purpose of ascertaining, on behalf of the Minister of Munitions, the position of the workings, actual and prospective, of any mines or abandoned mines through or near which it is proposed to sink any shaft or borehole for the purpose of searching for or getting petroleum, any officer appointed by the Director of the Geological Survey shall have the right to examine plans kept in pursuance of sections 20 or 21 of the coal mines act of 1911, or sections 14 or 19 of the metalliferous mines regulation act of 1872.

Any person entitled to sink a shaft or borehole for the purpose of searching shall give written notice to the Director of the Geological Survey and shall keep a record of any such shaft or borehole and shall allow the Director of the Geological Survey or any officer appointed by him free access to any such shaft and the privilege of inspecting all specimens obtained.

Records shall be kept and furnished to the Minister of Munitions of any petroleum obtained. Any one failing to keep and furnish any such record or any one making false statements shall be liable to a fine, not to exceed £50.

"Petroleum means all petroleum and its related hydrocarbons (except coal and bituminous shales and other stratified deposits from which oil can be extracted by distillation) and natural gas existing in its natural condition in strata."

SPECIAL PROVISIONS OF RECENT LEGISLATION

The mines (working facilities and support) act of 1923 (slightly amended by the mining industry act, 1926), provides that when there is danger that minerals may be left permanently unworked because they are comprised in or lie under copyhold land or land subject to a lease, exception, reservation, restriction, covenant, or condition, or otherwise not capable of being worked without the concurrence of two or more persons, or because minerals are owned in such small parcels that they can not be properly worked by themselves; when persons working adjoining boundaries with a view to reducing the amount of minerals to be left unworked or to enable the minerals to be worked more efficiently, and effect can not be given to the agreement because of the failure of the owners or lessors to concur; or when ancillary rights or privileges

are refused by persons with power to grant them, application may be made to the Board of Trade, which refers the application (if the case is considered a prima facie one) to the Railway and Canal Commission, which, upon being satisfied that the granting of the application is in the national interest, may issue an order, with certain terms and conditions. The Commission determines also the amount of compensation or consideration to be paid.

An order granted under these circumstances may confer rights on the tenant for life, and the rights shall be deemed to form part of the property, subject to the settlement of the estate of the deceased person or the property subject to a trust, as the case may be.

In the case of coal, in order that the terms of the mines (working facilities and support) act of 1923, may apply, it is necessary merely to prove the "national interest," and not personal interest in the minerals or danger of their being left unworked. (Pt. 2, mining industry act, 1926.)

The mines (working facilities and support) act of 1923, in section 8, provides that "if any person having an interest in any land is not entitled to support or sufficient support,[12] whether vertical or lateral, for any buildings or works, whether on or below the surface, erected or constructed, or intended to be erected or constructed, on or below the surface, and alleges that it is not reasonably practicable to obtain a right to such support by private arrangement . . . he may send to the Board of Trade an application that such restrictions may be imposed . . . as he may consider necessary to obtain sufficient support."

RENTS AND ROYALTIES, DUTIES, AND WELFARE LEVY

Rent and/or royalties are payable in accordance with the terms of the lease. There is no special taxation on mining enterprises, although a mineral rights duty (5 per cent) is payable on royalties, as well as a levy for a welfare fund.

Rent and Royalty

It is usual in mining leases to reserve both a fixed annual rent (otherwise known as a dead rent or minimum rent) and royalties, varying with the amount of minerals worked. The object of the fixed rent is to insure that the lessee will work the mine. If a fixed rent is reserved, it is payable until the expiration of the term, although the mine is not worked, or is exhausted during the currency of the term, or is not worth working, or is difficult or unprofitable to work because of faults or accidents. When a fixed rent is reserved, to commence from the time when a certain quantity of minerals shall have been obtained, and the lessee covenants to get that quantity without delay, the commencement of the payment shall not be delayed should the lessee fraudulently avoid completing the required production.

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12 - Right of support means the right to have the surface kept at its ancient and natural level; it is independent of the nature of the strata or the comparative value of the surface to the minerals, or to the fact that it is impossible from the nature of the ore to mark out pillars adequate to preserve support. (Halsbury's Laws of England, vol. 20, London, 1911, p. 571.)

Royalty, in the sense in which the word is used in connection with mining leases, is a payment to the lessor proportionate to the amount of the "demised" mineral worked within a certain period. Usually the royalties are made to merge into the fixed rent by means of a provision that the lessee may, without any additional payment, work (in each period for which a payment of fixed rent is made) so much of the minerals as would, at the royalties reserved, produce a sum equal to the fixed rent. Mining is necessarily speculative, and by the reservation of royalties the lessor benefits in proportion to the returns made.

Royalties take different forms. Sometimes a reservation is made of a share of the mineral worked, with or without the option of the lessor to require instead an equivalent in money. This form of reservation is most frequently adopted in leases of metallic minerals. Sometimes the reservation is made of a sum based upon a unit of mineral. Three different units are commonly made use of for this purpose, and the respective royalties are correspondingly known as acreage royalties, footage royalties, and tonnage royalties. In the case of acreage royalties, a unit is a superficial acre of the seem worked without regard to thickness. A footage acre is a species of acreage royalty, the unit being a superficial acre 1 foot thick. In the case of a tonnage royalty, a unit is a ton weight. As the lessor may reserve royalties in such form as he may consider most suitable or advantageous in any particular case, the questions that arise are questions of construction, and the decided cases can not be reduced to any principle generally applicable. Sometimes in colliery leases, coal consumed in working is free from royalty.

Mineral Rights Duty

The finance (1909-10) act of 1910, provides that a duty at the rate of 1 shilling for every 20 shillings (or 5 per cent) shall be charged, levied, and paid, for the financial year ending the 31st day of March, 1910, and every subsequent financial year, on the rental value of all rights to work minerals and of all mineral wayleaves.

The mineral rights duty shall not be charged with respect to common brick clay, common brick earth, or sand, chalk, limestone, or gravel.

Any lessor that pays any mineral rights duty and is himself a lessee of the right to work the minerals or of the wayleave with respect to which the duty is paid shall be entitled to deduct from the rent paid by him a sum equal to the mineral rights duty on a rental value of the same amount as the rent payable; and any person from whose rent any such deduction is made may make a similar deduction from any rent paid by him with respect to the right to work the minerals or with respect to the wayleave.

Royalties Welfare Levy

According to part 3, article 11, mining industry act of 1926, "every person liable to pay mineral rights duty on the rental value of rights to work coal and of mineral wayleaves in connection with coal, or who would be so liable but for any exception by common law or statute, shall be liable to pay for the financial year ending the 31st day of March, 1927, and for every

subsequent financial year a levy . . . at the rate in each case of 1 shilling for every 20 shillings of that rental value."

The provisions of the finance (1909-10) act of 1910, relating to the assessment, collection, and recovery of mineral rights duty and matters incidental thereto, shall apply to the royalties welfare levy; and in cases where the proprietor of the coal, or the person to whom rent is paid in respect to a right to work coal or a mineral wayleave in connection with coal, is liable to the payment of mineral rights duty, the royalties welfare levy shall be collected at the same time as and together with that duty, and in every other case the royalties welfare levy shall be payable on the first day of January in each year.

For the purpose of the welfare levy, "coal shall not include lignite or brown coal, but save as aforesaid shall include bituminous coal, cannel coal, anthracite, and all other minerals worked therewith other than minerals exempt from mineral rights duty."

Rental value shall be determined in accordance with the provisions of the finance (1909-10) act of 1910, relating to mineral rights duty, as amended by any subsequent enactment.

Article 12 of part 3 of the mining industry act, 1926, is concerned with the administration and collection of the welfare levy. It provides that no part of the proceeds of the royalties welfare levy shall be required to be allocated to any particular district.

MISCELLANEOUS DEFINITIONS[13]

"Rent" includes yearly or other rent; it is construed also as including any fine, premium, or foregift, and any payment, consideration, or benefit in the nature of a fine, premium, or foregift.

"Mineral wayleave" means any wayleave, air leave, water leave, or right to use a shaft granted to or enjoyed by a working lessee, whether above or below ground, for the purpose of access to or the conveyance of the minerals or the ventilation or drainage of his mine or otherwise, in connection with the working of the minerals.

"Total value" of minerals means the amount that the fee simple of the minerals, if sold in the open market by a willing seller, in their then condition, might be expected to realize.

"Capital value" of minerals means the total value, after allowing such deductions as the commissioners may allow for any works executed or expenditures of a capital nature incurred by or on behalf of any person interested in minerals for the purpose of bringing the minerals into working, or, where the minerals have been partly worked, such deduction as is in the opinion of the commissioners proportionate to the amount of minerals that have not been worked.

13 - Finance (1909-10) act, 1910: 10E7.ch.8.sec.23,24.

"Minerals" shall be treated as a separate parcel of land; but where the minerals are not comprised in a mining lease or being worked, they shall be treated as having no value as minerals, unless the proprietor of the minerals, in his return furnished to the commissioners, specifies the nature of the minerals and his estimate of their capital value. Minerals that are comprised in a mining lease or are being worked shall be treated as a separate parcel of land for the purpose of assessment of duty.

I.C.6517.
September, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

FATALITIES IN TENNESSEE COAL MINES[1]

By H. B. Humphrey[2] and F. E. Cash[3]

EXTENT AND PURPOSE OF REPORT

The Southern Appalachian coal field extends southward across Tennessee in a belt about 50 miles wide between the eastern and middle sections of the State. The earliest record of the production of coal in Tennessee is for 1840, and the first inspection laws were passed in 1881. In 1887, a second act of the legislature created the office of Chief Inspector, but little was done until 1891 when a new act was passed giving to the Department of Mines added duties and powers. Since that time, except for the years 1915 to 1918, comprehensive annual reports have been published. Data for the tables given in this paper are taken from these annual reports, from those of the Southern Appalachian Coal Operators' Association, and from the publications of the United States Bureau of Mines, which has maintained a station at Jellico, Tenn., since 1911. Complete figures for 1930 are not available yet and they are not included in the averages. The State Department of Mines, the operating companies, and the United States Bureau of Mines cooperate in accident-prevention work.

It is the purpose of this report to present in uniform tables the available statistics in regard to fatal accidents in Tennessee and in the coal mines of the United States, in order that the progress in mine safety in Tennessee may be compared with that of the country as a whole, and the present period may be compared with various periods of the past, to indicate the main points of attack in the reduction of accidents.

ACKNOWLEDGMENT

The state records were made available through the courtesy of the chief inspector, A. W. Evans.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6517."

2 Assistant engineer, U. S. Bureau of Mines, Safety Station, Birmingham, Ala.

3 District engineer, U. S. Bureau of Mines Safety Station, Birmingham, Ala.

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FATALITY RATES

Table 1 lists the fatality rates for Tennessee and for the United States for the period 1891 through 1929 (figures for 1930 are not final). The deaths per million tons mined is an economic or cost factor which depends largely on efficiency of mining methods and labor, as well as on favorable or unfavorable natural conditions. The fatalities per million man-hours is a gage of the safety of the worker while at his work, and is unquestionably the fairest basis of comparison of accidents between different fields and different industries. The major disasters in Tennessee coal mines in 1895, 1901, 1902, 1911, and 1926 are responsible for the high rates of those years. Comparison of the rates for different periods of years given at the end of Table 1 shows a much higher rate for men killed per million tons of coal mined in Tennessee than in the United States. For the period 1891-99 the respective rates were 7.31 and 5.59; 1900:1909, 9.35 and 5.78; 1910-19, 5.40 and 4.53; and 1920-29, 4.23 and 3.82. A disaster during 1926 kept the averages for 1925 to 1929 above that of the United States. The rates for 1927 and 1928 are better than those of the United States, and in 1929 the death rate per million tons mined was 3.71 for Tennessee and 3.59 for the whole country. For the entire period 1891-1929, the respective rates were 6.31 and 4.65, showing that Tennessee's coal-mine safety record on a tonnage basis is by no means as good as the average of the coal mines of the United States.

The true measure of the hazard to the miner - men killed per million man-hours of exposure - shows Tennessee to have a slightly lower average than that of the United States, except for the years 1900 to 1909 during which period some of the worst disasters occurred. In 1891 to 1899 Tennessee had a rate of 1.47 men killed per million man-hours compared with 1.70 in the United States; in 1910 to 1919, 1.53 to 1.75; in 1920 to 1929, 1.56 to 1.83; and in 1925 to 1929, 1.80 to 1.85. For the entire period, Tennessee had a rate of 1.75 and the United States 1.83; the last three years give the former an average of 1.27 against 1.84; hence, on the basis of man-hours of exposure, Tennessee's coal mines have a slightly better safety record than the average for the coal mines of the United States as a whole, and the Tennessee coal-mine safety record on this basis was much better during the past three years than the record of the United States as a whole in coal mining.

Table 2 gives the production, man-hours, and tons per man-hour in the United States and in Tennessee for each year from 1891 to 1929, and for 10-year periods. Tennessee is much behind the remainder of the country in labor efficiency because of the general smaller thickness of beds and size of mine, and because of the irregular development and operation made possible by their being drift mines, so that work could be started and stopped at different points with small expense.

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Table 1.- Fatality rates in Tennessee and the United States since 1891, according to State and Federal statistics

Year	Killed per million tons		Killed per million man-hours	
	United States	Tennessee	United States	Tennessee
1891	6.06	9.13	1.68	1.83
1892	5.98	6.71	1.74	1.17
1893	5.39	6.83	1.57	1.18
1894	5.91	6.41	1.74	1.27
1895	6.00	14.67	1.83	3.23
1896	5.85	8.25	1.78	1.75
1897	5.07	3.47	1.65	0.75
1898	4.97	6.17	1.67	1.21
1899	5.08	5.38	1.72	1.14
1900	5.72	2.56	1.91	0.50
1901	5.40	11.61	1.80	2.37
1902	5.81	53.40	2.06	12.02
1903	5.47	5.41	1.75	1.26
1904	5.88	5.76	1.94	1.35
1905	5.78	5.22	2.01	1.18
1906	5.27	5.27	1.88	1.35
1907	6.78	4.46	2.65	1.21
1908	5.97	5.61	2.00	1.50
1909	5.73	4.98	2.20	1.35
1910	5.62	5.49	2.15	1.53
1911	5.35	17.21	2.02	4.61
1912	4.53	2.73	1.84	0.79
1913	4.89	5.19	2.07	1.39
1914	4.78	4.37	1.78	1.25
1915	4.27	4.37	1.69	1.37
1916	3.77	2.44	1.53	0.73
1917	4.14	4.54	1.75	1.25
1918	3.80	2.04	1.61	0.60
1919	4.19	5.18	1.77	1.48
1920	3.45	3.03	1.57	0.97
1921	3.94	3.23	1.74	1.09
1922	4.16	5.00	2.02	1.73
1923	3.74	3.57	1.82	1.01
1024	4.20	4.43	1.99	1.83
1925	3.84	4.83	1.93	1.82
1926	3.83	8.10	1.86	3.32
1927	3.72	2.86	1.82	1.15
1928	3.78	3.43	1.92	1.24
1929	3.59	3.71	1.78	1.35
1930	3.79	4.00	1.79	1.29
1891-1929	4.65	6.31	1.83	1.75
1891-1899	5.59	7.31	1.70	1.47
1900-1909	5.78	9.35	2.03	2.24
1910-1919	4.53	5.40	1.75	1.53
1920-1929	3.82	4.23	1.83	1.56
1925-1929	3.76	4.62	1.85	1.80
1900-1929	4.56	6.22	1.86	1.80

Table 2.- Coal production and labor in United States and Tennessee coal mines since 1891

Year	Tonnage		Man-hours		Tons per man-hour	
	United States	Tennessee	United States	Tennessee	United States	Tennessee
1891	160,000,000	2,413,678	570,000,000	12,000,000	0.28	0.20
1892	166,000,000	2,092,064	570,000,000	12,000,000	0.29	0.17
1893	178,000,000	1,902,258	610,000,000	11,000,000	0.29	0.18
1894	165,000,000	2,180,879	550,000,000	11,000,000	0.30	0.19
1895	191,000,000	2,535,644	625,000,000	11,500,000	0.31	0.22
1896	186,000,000	2,663,106	610,000,000	12,500,000	0.31	0.21
1897	195,000,000	2,888,840	600,000,000	13,500,000	0.32	0.21
1898	214,000,000	3,084,748	640,000,000	15,640,000	0.33	0.20
1899	245,000,000	3,736,134	725,000,000	17,676,000	0.34	0.21
1900	265,000,000	3,904,048	778,500,000	20,000,000	0.39	0.20
1901	290,000,000	3,785,672	865,500,000	18,500,000	0.34	0.21
1902	300,000,000	4,232,332	918,400,000	18,830,000	0.33	0.22
1903	350,000,000	4,810,758	1,002,000,000	21,123,000	0.35	0.23
1904	345,000,000	4,847,242	1,032,000,000	20,826,000	0.34	0.23
1905	390,000,000	5,552,576	1,108,000,000	24,517,000	0.35	0.23
1906	410,000,000	6,272,457	1,135,000,000	24,470,000	0.36	0.26
1907	475,000,000	6,940,911	1,215,000,000	25,586,000	0.39	0.27
1908	410,000,000	6,082,851	1,222,000,000	22,560,000	0.33	0.27
1909	460,800,000	6,207,483	1,200,000,000	23,000,000	0.38	0.27
1910	501,596,378	6,908,688	1,305,000,000	24,820,000	0.38	0.28
1911	496,371,126	6,466,224	1,311,000,000	24,062,000	0.38	0.27
1912	534,466,580	6,578,754	1,311,000,000	22,568,000	0.41	0.29
1913	569,960,219	6,739,486	1,346,000,000	25,204,000	0.42	0.27
1914	513,525,477	5,943,258	1,378,437,219	20,665,000	0.40	0.29
1915	531,619,487	5,730,361	1,339,278,683	18,259,000	0.40	0.31
1916	590,098,175	6,137,449	1,452,788,095	20,565,000	0.41	0.30
1917	651,402,374	6,194,221	1,575,863,591	22,340,000	0.42	0.28
1918	678,211,904	6,831,048	1,599,853,614	23,509,000	0.42	0.29
1919	553,952,259	5,387,606	1,309,154,891	18,968,000	0.42	0.28
1920	658,264,932	6,921,848	1,451,162,000	21,560,000	0.45	0.32
1921	506,395,401	4,344,564	1,145,738,345	12,804,000	0.44	0.34
1922	476,951,121	5,209,911	979,994,880	15,016,000	0.48	0.35
1923	657,903,671	6,170,863	1,356,088,890	16,675,000	0.48	0.37
1924	571,613,400	4,750,754	1,207,474,560	11,992,000	0.48	0.40
1925	581,869,890	5,596,997	1,160,333,952	14,276,000	0.50	0.39
1926	657,804,437	6,089,162	1,352,840,229	15,015,000	0.48	0.40
1927	597,859,000	5,935,130	1,219,079,373	14,761,000	0.49	0.40
1928	576,093,000	5,545,762	1,135,542,452	14,512,000	0.51	0.39
1929	608,816,000	5,663,155	1,225,000,000	15,550,000	0.50	0.36
1930	531,432,000	5,103,000	(1)	(1)	(1)	(1)
1891-1929	16,891,038,000	195,278,922	42,639,000,000	709,046,000	0.40	0.28
1900-1929	15,919,038,000	171,781,571	37,139,000,000	592,230,000	0.41	0.29
1891-1899	1,700,000,000	23,497,351	5,500,000,000	116,816,000	0.31	0.20
1900-1909	3,696,000,000	52,636,330	10,477,000,000	219,412,000	0.35	0.24
1910-1919	5,571,204,000	62,917,095	14,429,000,000	220,960,000	0.39	0.28
1920-1929	5,873,442,525	56,228,146	12,233,255,000	151,850,000	0.48	0.37
1925-1929	3,002,618,000	28,830,206	6,092,796,000	74,115,000	0.49	0.39

(1) Data not available.

Table 3 is a summary of the 10-year periods giving the production, fatalities, and tons per man killed in Tennessee and the United States. This is an inverted form of the rate per million tons mined, but is given as a common standard of comparison. The average tonnage per man killed in the United States from 1891 to 1929 was 215,000; in Tennessee it was 158,545. The United States shows an almost constant improvement from 182,000 tons per fatality in 1891-99 to 261,660 tons in 1920-29. In Tennessee, the figure varied from a low of 107,004 in 1900-1909 to 237,250 tons in 1920-29. The highest average for the United States was 290,000 tons in 1920, and for Tennessee was 487,932 tons in 1918. The lowest average for the United States was 144,000 tons in 1907, and for Tennessee was 18,727 tons in 1902. In 40 years (1891-1930) 80,463 fatal accidents were recorded in the coal mines of the United States; 1,258 of them in Tennessee. During 1891-1929, Tennessee produced 1.1 per cent of the coal in 1.6 per cent of the man-hours, with 1.6 per cent of the fatal accidents. The production per man-hour is 30 per cent below the average of the United States. In 1929, Tennessee produced 0.9 per cent of the coal in 1.2 per cent of the man-hours with 0.9 per cent of the fatal accidents. This is not the best record for any year, but it is at least better than the average. (See Table 5, following.)

Table 3.- Coal mined per man killed in the United States and Tennessee by 10-year periods

	Coal mined, tons		Men killed		Tons per man killed	
Period	United States	Tennessee	United States	Tennessee	United States	Tennessee
1891-1899	1,700,000,000	23,497,351	9,381	171	182,000	137,411
1900-1909	3,696,000,000	52,636,330	21,391	492	173,000	107,004
1910-1919	5,621,300,000	62,917,095	25,229	338	223,000	186,145
1920-1929	5,873,746,575	56,228,146	22,448	237	261,660	237,250
1925-1929	3,002,618,000	28,830,206	11,333	133	264,944	216,765
1927	597,859,000	5,935,130	2,224	17	268,821	249,125
1928	576,093,000	5,545,762	2,176	19	264,702	291,882
1929	608,992,000	5,663,155	2,181	21	279,226	269,676
1900-1929	15,191,038,000	171,781,571	69,068	1,067	220,000	160,995
1891-1929	16,891,038,000	195,278,922	78,449	1,238	215,000	158,545
1930	531,432,000	5,103,000	2,014	20	264,000	255,000

The low fatality rates on the basis of time exposure, which are marred here and there by explosion disasters, may be attributed to a good grade of steady native white labor which comprises the major part of the mine crews. Added to this have been the work of the State Mine Inspectors to induce safer practices, and of the Southern Appalachian Coal Operators' Association to reduce accidents in cooperation with the United States Bureau of Mines. Effective work toward safer and better practices is accomplished through a division of this association known as the Southern Appalachian Efficiency Association. The practices at some mines have been extremely unsafe, whereas others have to a large extent used practically all reasonable precautions.

Table 4 has the tonnage and time exposure ratings for five divisions of Tennessee mines for the years 1927, 1928, and 1929. The mines were divided into these five classes on the basis of yearly production, and the rates calculated for the 3-year average to determine what difference in hazards there might be between mines of different production capacity, as shown by the fatal accidents.

Class I is made up of mines producing 10,000 tons or less a year; the number of openings is decreasing--from 32 in 1927 to 18 in 1929. Only two fatal accidents were recorded, both in 1929; for the 3-year period this gives a high fatality rate for the tonnage produced, but an average rate for the time worked of 1.20 per million man-hours; this shows low efficiency in labor, high accident cost per ton, but average safety for the individual.

Class II, mines producing 10,000 to 50,000 tons a year, increased in number from 17 in 1927 to 28 in 1929, with double the tonnage and working hours. Five fatal accidents were recorded in 1927, none in 1928, and five in 1929; for the three years the average rate per million tons was 5.22 and 1.52 per million man-hours. The labor efficiency in this group is about double that of class I, yet it is only 75 per cent of that in the larger mines. Probably, safety receives less attention in this class of mines than in the others.

Class III, mines producing 50,000 to 100,000 tons a year, has the best record. With 15 mines in 1927, 17 in 1928, and 19 in 1929, only 2.10 fatalities are recorded per million tons and 0.88 per million man-hours. Evidently, this size of mine is favorable to safe and efficient operation.

Class IV, mines producing 100,000 to 200,000 tons a year, declined from 16 mines in 1927 to 11 in 1928 and 1929. Their record is slightly better than the average of all, being 3.13 per million tons and 1.17 per million man-hours.

Class V, comprising the largest mines in the State, numbered 8 in 1927, 9 in 1928, and only 6 in 1929. Their efficiency was good in 1927 and 1928, somewhat lower in 1929, but still ahead of the other groups. Their fatal accidents increased from 6 in 1927 to 7 in 1928 and to 8 in 1929, with fewer mines in 1929 than in 1927 or 1928, and fewer mines in 1927 than in 1928.

For all mines both ratings are higher in 1928 than in 1927, and still higher in 1929.

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Table 4.- Fatality rates in Tennessee mines during 1927, 1928, and 1929

Year	Mines	Tons	Hours	Killed	Rate per million Tons	Rate per million Man-hours	Tons per man-hour
CLASS I - UNDER 10,000 TONS							
1927	32	91,135	650,000	0			0.14
1928	23	75,147	638,000	0			0.12
1929	18	59,391	383,000	2			0.15
3 years		225,673	1,671,000	2	9.00	1.20	0.14
CLASS II - 10,000 TO 50,000 TONS							
1927	17	435,754	1,399,000	5			0.31
1928	24	605,549	2,118,000	0			0.29
1929	28	877,860	2,958,000	5			0.29
3 years		1,919,163	6,575,000	10	5.22	1.52	0.29
CLASS III - 50,000 TO 100,000 TONS							
1927	15	1,074,159	2,730,000	1			0.37
1928	17	1,236,054	2,555,000	5			0.48
1929	19	1,508,060	3,821,000	2			0.40
3 years		3,818,273	9,106,000	8	2.10	0.88	0.42
CLASS IV - 100,000 TO 200,000 TONS							
1927	16	2,234,800	5,322,000	5			0.42
1928	11	1,324,255	4,025,000	7			0.33
1929	11	1,545,253	4,334,000	4			0.36
3 years		5,104,308	13,681,000	16	3.13	1.17	0.37
CLASS V - 200,000 TO 500,000 TONS							
1927	8	2,099,282	4,600,000	6			0.46
1928	9	2,304,757	4,864,000	7			0.47
1929	6	1,672,591	4,054,000	8			0.41
3 years		6,076,630	13,518,000	21	3.43	1.55	0.45
TOTALS - ALL MINES							
1927	88	5,935,130	14,701,000	17	2.86	1.16	0.40
1928	84	5,545,762	14,260,000	19	3.43	1.33	0.39
1929	82	5,663,155	15,550,000	21	3.71	1.35	0.36
3 years		17,144,047	44,511,000	57	3.33	1.28	
1930		5,103,000	(1)	20	(1)	(1)	(1)

(1) Data not available.

CAUSES OF FATAL ACCIDENTS

Table 5 is a compilation of the fatal accidents in Tennessee since 1891 by cause and giving the yearly total for the United States. For the years in which the State mine inspectors' reports were published the total of nonfatal accidents reported in Tennessee is also given; these State reports and bulletins of the United States Bureau of Mines were used as the basis for making up this table.

[illegible]

Before 1904, the nonfatal accidents reported were about three times the fatalities, indicating that a greater number of accidents were fatal in those days, or more probably that the nonfatal accidents were not as completely reported as they were later. Since that time the proportion appears to be about 5 to 1. These are lost-time reportable accidents.

Table 5.- Yearly fatalities in Tennessee by causes

Year	Falls	Haulage	Electricity	Machinery	Explosives	Gas and dust	Miscellaneous	Total		Reported nonfatal,
								Tennessee	United States	Tennessee
1891	13	3	–	–	4	2	–	22	956	45
1892	11	1	–	–	3	–	–	14	991	15
1893	6	4	–	–	1	–	2	13	958	58
1894	12	1	–	1	–	–	–	14	958	30
1895	7	–	–	–	2	28	–	37	1,142	111
1896	18	1	–	–	3	–	–	22	1,083	25
1897	10	–	–	–	–	–	–	10	990	35
1898	14	2	–	–	1	1	–	19	1,062	77
1899	13	4	–	–	1	1	–	20	1,241	74
1900	9	–	–	–	1	–	–	10	1,492	33
1901	17	4	–	1	1	20	1	44	1,549	89
1902	18	5	–	–	3	200	–	226	1,895	103
1903	15	8	1	–	2	–	–	26	1,752	70
1904	19	4	–	–	1	4	–	28	2,004	129
1905	18	3	1	–	7	–	–	29	2,232	147
1906	20	6	–	–	2	1	4	33	2,138	210
1907	20	4	2	–	1	3	1	31	3,242	195
1908	27	2	4	–	1	–	–	34	2,445	195
1909	18	5	2	1	2	1	2	31	2,642	197
1910	22	4	2	–	7	3	–	38	2,821	210
1911	14	8	4	–	1	84	–	111	2,656	198
1912	14	3	–	–	1	–	–	18	2,419	118
1913	26	6	1	–	–	1	1	35	2,785	119
1914	21	3	–	–	1	1	–	26	2,454	129
1915	15	4	2	–	3	1	–	25	2,269	–
1916	9	3	3	–	–	–	–	15	2,226	–
1917	12	2	1	1	–	12	–	28	2,696	–
1918	10	3	1	–	–	–	–	14	2,580	–
1919	18	7	–	–	1	2	–	28	2,323	150
1920	15	5	–	–	–	1	–	21	2,272	175
1921	8	3	1	1	1	–	–	14	1,995	125
1922	17	3	2	1	–	1	2	26	1,984	110
1923	9	7	1	–	2	3	–	22	2,462	175
1924	7	5	2	–	2	3	2	21	2,402	160
1925	10	3	–	–	1	13	–	27	2,234	210
1926	14	4	1	–	3	27	–	49	2,518	250
1927	10	3	3	1	–	–	–	17	2,224	200
1928	9	5	1	–	4	–	–	19	2,176	200
1929	15	5	1	–	–	–	–	21	2,181	125
1930	16	3	–	–	1	–	–	20	2,014	130
1891-1929	560	143	36	7	63	413	16	1,238	78,449	–
1900-1929	456	127	36	6	48	381	13	1,067	69,068	–

From 1891 to 1929, the division of fatal accidents according to cause has been as given in Table 6.

Table 6.- Fatal accidents in Tennessee by causes

Cause	Tennessee Past 39 years, per cent	Tennessee Past 3 years, per cent	United States 10 years, per cent
Falls of rock and coal	45.3	59.5	52.0
Haulage	11.6	22.8	19.0
Electricity	2.9	8.8	5.2
Machinery	0.5	1.8	1.9
Explosives	5.1	7.1	5.3
Gas and dust explosions	33.3	0.0	11.9
Miscellaneous	1.3	0.0	4.7
Total	100.0	100.0	100.0

Discussion of Causes

Falls of rock and coal in Tennessee bear essentially the same relation to the total as exists for the whole United States; the high percentage of the last three years was due to the absence of explosion fatalities, leaving the others larger in respect to the total. Workers at an average of 14 per year have been killed by falls of rock and coal since 1891, so that any number of fatalities below this may be considered an improvement. This is a form of accident which can be reduced by strict inspection, scaling of loose material, and timbering, with constant supervision by foremen. The deaths from haulage average four a year, and are now approximately 20 per cent of the total. Electricity causes about 8 per cent of the fatalities, with the prospect of inducing a greater number as more power is used in the mines. Explosives account for about 5 per cent, which is on a par with the remainder of the country. The average is about two a year--much too high now that 65 per cent of the explosives used are of permissible type and not black powder as before 1906, and also now that the coal is being undercut before shooting. As the remainder of the accidents of this type are due to improper handling and carelessness, they can and should be eliminated. Over the whole period from 1891, explosions have been responsible for one-third of the deaths in Tennessee coal mining. The average for the United States for the last 10 years is less than 12 per cent. Tennessee lost 27 lives in an explosion in 1926, but has had a "clean slate" since that year. Only a few of the mines in Tennessee are gassy; many are naturally wet, so that with reasonable care in the use of electricity and proper precautions in ventilation and blasting, explosions in the future can be made a rarity. The use of rock-dust for safety and as an aid to illumination where not practised should be given serious consideration, because widespread explosions are preventable, and rock-dusting together with proper ventilation is one of the chief means of prevention.

In Table 7, fatal accidents in Tennessee for the years 1927, 1928, and 1929 are classified by causes and according to the size of the mine in which they occurred. Mines in Class I, with production of less than 10,000 tons per year, have the highest percentage of fatalities in proportion to the percentage of tonnage mined; this is in accord with data from other States also. Class III mines, with annual production of 50,000 to 100,000 tons,

have the most favorable records. Rock falls were responsible for both fatalities in Class I, for 8 of 10 fatalities in Class II, 3 of 8 in Class III, 12 of 16 in Class IV, and for 9 of 21 in Class V. This means that falls are the major source of accidents in all mines of this State, and that in the larger mines causes are more varied than in the smaller mines. Among these other causes is haulage, which is charged with four deaths in Class II (all of these deaths occurred at one time, and this, of course, is not a general condition), but with seven in Class V; these fatalities point the way to a needed reduction of haulage accidents through more general observance of recognized safe practice in haulage. The more progressive mines in Tennessee have efficient haulage with relatively few accidents. From the record of the last three years the danger from electrocutions lies in Class IV mines, which accounted for three accidents of five of this type for all classes. As the use of electric power is extended, electrocutions will increases unless proper safeguards are provided as part of the installation and use of electricity, with no toleration of temporary "layouts," or of any but the safest practices. The four deaths charged to explosives in 1928 were caused at one time by careless handling of black powder, which has no place in safe coal mining.

Table 7.- Causes of fatalities in Tennessee by classes of mines during 1927, 1928, and 1929

Year	Rock falls	Haul-age	Elec-tricity	Machinery	Explo-sives	Gas and dust	Miscel-laneous	Total	Mines	Tonnage, per cent	Fatal, per cent
CLASS I - UNDER 10,000 TONS											
1927	-	-	-	-	-	-	-	0	32	1.5	0.0
1928	-	-	-	-	-	-	-	0	23	1.4	0.0
1929	2	-	-	-	-	-	-	2	18	1.1	9.5
1930	1	-	-	-	-	-	-	1	-	-	-
Average	-	-	-	-	-	-	-	1	-	1.3	3.2
CLASS II - 10,000 TO 50,000 TONS											
1927	4	-	1	-	-	-	-	5	17	7.4	29.4
1928	-	-	-	-	-	-	-	0	24	10.8	0.0
1929	4	1	-	-	-	-	-	5	28	15.5	23.8
1930	1	-	-	-	-	-	-	1	-	-	-
Average	-	-	-	-	-	-	-	-	-	11.2	14.4
CLASS III - 50,000 TO 100.000 TONS											
1927	1	-	-	-	-	-	-	1	15	18.2	5.9
1928	-	4	1	-	-	-	-	5	17	22.2	26.4
1929	2	-	-	-	-	-	-	2	19	26.7	9.5
1930	2	-	-	-	-	-	-	2	-	-	-
Average	-	-	-	-	-	-	-	-	-	22.3	13.9
CLASS IV - 100,000 TO 200,000 TONS											
1927	2	1	2	-	-	-	-	5	16	37.5	29.4
1928	7	-	-	-	-	-	-	7	11	23.7	36.8
1929	3	-	1	-	-	-	-	4	11	27.1	19.1
1930	9	-	-	-	-	-	-	9	-	-	-
Average	-	-	-	-	-	-	-	-	-	29.4	28.4
CLASS V - 200,000 TO 500,000 TONS											
1927	3	2	-	1	-	-	-	6	8	35.3	35.3
1928	2	1	-	-	4	-	-	7	9	41.9	36.8
1929	4	4	-	-	-	-	-	8	6	29.6	38.1
1930	3	-	-	-	-	-	-	3	-	-	-
Average	-	-	-	-	-	-	-	-	-	35.6	36.7
ALL MINES											
1927	10	3	3	1	-	-	-	17	88	100.0	100.0
1928	9	5	1	-	4	-	-	19	84	100.0	100.0
1929	15	5	1	-	-	-	-	21	82	100.0	100.0
1930	16	3	-	-	1	-	-	20			

NONFATAL ACCIDENTS IN 1930

During 1930, 490 lost-time accidents were reported to the Southern Appalachian Coal Operators' Association, compared with 690 in 1929. Table 8 lists 130 of these accidents, which include everything from a mashed finger to a broken leg and are probably a representative group of those of which a record was obtained.

Fifty-six of these lost-time accidents were from falls of slate and coal, 50 of which were sustained by miners and loaders; the remainder was scattered among all classes of workers. This gives an indicated proportion of 90 per cent of falls happening at the face.

Haulage is charged with 36 of the total of 130 lost-time accidents. Nineteen were accidents to miners and loaders, 3 to motormen, and 11 to couplers. The miners and loaders are injured in handling cars from the entry to the face, due chiefly to low top and insufficient clearance. A coupler's job is a hazardous one under present haulage practice, where work is demanded at high speed and usually with poor track, equipment, clearance, and lighting.

There were 10 accidents from mining machines and three from explosives. It would seem that accidents of these two classes are due chiefly to carelessness and could be prevented by proper care and rigid efficient supervision.

Many lost time injuries from tools were reported--16 out of 130 such accidents; the explanation seems to be that many eye injuries occurred from picking rock and coal, and many persons were hurt by striking the hand or foot with ax or pick. Goggles can be used to prevent the eye injuries, and the hand and foot injuries can at least be materially decreased in number and severity by careful inspection and instruction and by the wearing of safety shoes.

Table 8.- Reported non-fatal accidents in Tennessee during 1930

Occupation	Falls	Haulage	Electricity	Machinery	Explosives	Gas and dust	Tools	Miscellaneous	Total
Miner and loader	50	19	0	1	3	0	10	1	84
Machine man	1	0	1	4	0	0	1	0	7
Machine helper	1	0	0	4	0	0	0	1	6
Motorman	0	3	0	0	0	0	0	0	3
Coupler	0	11	0	0	0	0	0	0	11
Trackman	1	1	0	0	0	0	2	1	5
Driver	1	0	0	0	0	0	1	1	3
Pumpman	0	0	0	0	0	0	1	1	2
Mechanic	1	0	0	1	0	0	1	1	4
Car trimmer	0	1	0	0	0	0	0	2	3
Face boss	0	1	0	0	0	0	0	0	1
Safety man	1	0	0	0	0	0	0	0	1
Total	56	36	1	10	3	0	16	8	130

CONCLUSION

Apparently, operations in Tennessee range from poorly managed properties with extremely unsafe practices, many of which somehow escape accident, to well-regulated and reasonably safe mines. Favored as they are by natural conditions, the men in charge of the mines in this State can make a fatal accident a rare occurrence and other types of accidents relatively few by the installation of safe and efficient equipment, and insistence on its proper upkeep and use. By such means the cost of production is lowered and the worker is protected in the enjoyment of his life and health.

I.C. 6519
October, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

FATAL ACCIDENTS IN ALABAMA COAL MINES DURING 1930[1]

By F. E. Cash[2] and H. B. Humphrey[3]

Sixty-one fatal accidents were reported in the mines of Alabama during 1930. This number is a decrease of 15 per cent as compared with 1929; however, production was 13 per cent and the man-hours worked were 16 per cent less than in the previous year. There were 21,731 men employed inside and 3,386 outside.

Table 1 gives the comparative rates per million tons and per million man-hours for the United States and for Alabama for 1929 and 1930. These rates are based on figures published by the State and by the United States Bureau of Mines, with preliminary figures for 1930. (These sources of data are used throughout this paper.)

Table 1. - Fatality rates in Alabama during 1930

Year	Deaths per million tons		Deaths per million man-hours worked	
	United States	Alabama	United States	Alabama
1929	3.59	3.91	1.87	1.40
1930	3.80	3.80	1.96	1.40
Average for 1926-1930	3.76	4.57	1.87	1.63

The fatality rate based on production was 3.91 per million tons in 1929 and 3.80 in 1930 for Alabama whereas for the United States the rate rose from 3.59 in 1929, to 3.80 in 1930; the improvement in Alabama was due to a decrease in fatalities and possibly to increased efficiency of labor through the growing use of mechanical mining methods. The rate for the United States was increased, largely because of a decrease of 13 per cent in production and a decrease in fatalities of only 8 per cent by reason of a large increase in deaths from explosions and roof falls. Table 2 shows how much coal was mined and how many hours of work were performed during the past two years.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6519."

2 - District engineer, U. S. Bureau of Mines Safety Station, Birmingham, Ala.

3 - Assistant engineer, U. S. Bureau of Mines Safety Station, Birmingham, Ala.

Table 2. - Production and labor in Alabama during 1930

Year	Coal, tons		Man-hours		Tons per man-hour	
	United States	Alabama	United States	Alabama	United States	Alabama
1929	608,817,000	18,416,000	1,225,000,000	51,683,000	0.50	0.36
1930	531,432,000	16,100,000	1,025,000,000	43,423,000	0.52	0.37
Average for 1926-1930	590,436,000	18,856,000	1,191,600,000	53,220,000	0.50	0.36

The tonnage mined per fatality is given in Table 3.

Table 3. - Tons of coal mined per fatality in Alabama during 1930

Year	Men killed		Tons per man killed	
	United States	Alabama	United States	Alabama
1929	2187	72	277,920	255,778
1930	2015	61	263,730	262,000
Average, 1926-1930	2223	86	53,126	43,648

In 1930, Alabama produced about 3 per cent of the coal mined in the United States and had about 3 per cent of the fatalities. Some excellent records were made and maintained; some mines produced one-half to three-quarters of a million tons of coal without a fatality.

Table 4 gives the fatality rates by classes of mines -- that is, grouped by tonnage.

Table 4. - Fatality rates in Alabama coal mines during 1930

Class	Year	Mines	Tons	Man-hours	Fatalities	Fatalities per million Tons	Fatalities per million Man-hours
Class I, under 10,000 tons	1929	84	163,207	810,000	2	12.25	2.47
	1930	93	164,000	776,000	3	18,30	3.88
Class II, 10,000 to 50,000 tons	1929	37	847,871	2,793,000	13	15.35	4.64
	1930	26	660,000	2,148,000	3	4.54	1.39
Class III, 50,000 to 100,000 tons	1929	33	2,441,950	7,345,000	10	4.09	1.36
	1930	36	2,627,000	7,870,000	19	7.22	2.41
Class IV, 100,000 to 200,000 tons	1929	27	3,674,775	11,255,000	12	3.27	1.06
	1930	22	3,123,000	8,643,000	12	3.84	1.38
Class V, 200,000 to 500,000 tons	1929	20	6,156,877	17,618,000	25	4.06	1.42
	1930	17	5,395,300	13,328,000	17	3.19	1.28
Class VI, Over 500,000 tons	1929	7	5,131,234	11,862,000	10	1.95	0.84
	1930	6	4,129,700	10,658,000	7	1.70	0.66
All mines	1929	208	18,416,000	51,683,000	72	3.91	1.40
	1930	200	16,100,000	43,423,000	61	3.80	1.40

Fatalities by causes are listed in Table 5.

Table 5. - Causes of fatalities during 1930

Cause	1928 and 1929		1930	
	Number	Per cent	Number	Per cent
Falls of rock and coal ..	28	40.0	34	55.7
Haulage	19	26.0	5	8.2
Electricity	11	17.3	11	18.0
Machinery	0	2.9	2	3.3
Explosives	0	2.9	1	1.7
Gas and dust explosions .	12	8.7	8	13.1
Miscellaneous	2	2.2	0	0.0
Total	72	100.0	61	100.0

ANALYSIS OF FATALITIES

In Table 6 the fatal accidents in Alabama will be given for the six divisions as in Table 5 according to cause. Falls of rock and coal increased from 28 to 34; Class I mines had three accidents for falls compared with one in 1929, due chiefly to poorly set and insufficient timber and to lack of supervision. Class II mines had two fatal accidents in dangerous pillar work. Class III mines had 10 fatal accidents in 1930 and five in 1929; of these, 5 were caused by working under loose rock with little or no inspection, 4 were due largely to insufficient timber, and 1 occurred at a high pillar face. Class IV mines had only two fatalities in 1930, the same number as in 1929; both resulted from working under loose rock. In Class V mines there were 12 fatal accidents in 1930 from rock falls as compared to 10 in 1929; 5 deaths resulted from falls at the face where the miner was working under overhanging coal or rock, 2 resulted from falls while working on pillars where men had been forbidden to work, 2 from falls of loose scale in manways, 1 from falls due to insufficient timber, and 2 from falls when props were being knocked out in moving a conveyor and a machine. Class VI mines had seven fatal accidents in 1929 and five in 1930; three in 1930 were due to failure to examine the roof before working and two to insufficient timber.

The increase in fatalities from falls of roof in Alabama's coal mines in 1930 over 1929 may be charged to a lack of care in two mines of Class V and to three mines in Class III, in one of which the use of safety props has since been adopted; one fatality was due to dangerous pillar work, and one to insufficient care in testing roof.

Table 6 lists the fatalities by causes and by class of mine.

Table.6. - Causes of coal-mine fatalities in Alabama during 1930

Class	Year	Rock fall	Haul-age	Elec-tricity	Ma-chin-ery	Explo-sives	Gas and dust explo-sives	Miscel-laneous	Total	Ton-nage, per cent	Fatal-ities, per cent
Class I - Under 10,000 tons	1929	1	1	0	0	0	0	0	2	0.9	2.8
	1930	3	0	0	0	0	0	0	3	1.0	4.8
Class II - 10,000 to 50,000 tons	1929	3	0	0	0	0	10	0	13	4.6	18.1
	1930	2	0	0	1	0	0	0	3	4.1	5.0
Class III - 50,000 to 100,000 tons	1929	5	2	3	0	0	0	0	10	13.2	13.8
	1930	10	0	2	0	0	7	0	19	16.3	31.1
Class IV - 100,000 to 200,000 tons	1929	2	5	4	0	0	0	1	12	20.0	16.7
	1930	2	2	7	0	0	1	0	12	19.4	19.7
Class V - 200,000 to 500,000 tons	1929	10	9	3	0	0	2	1	25	33.3	34.8
	1930	12	3	1	0	1	0	0	17	33.5	27.9
Class VI - Over 500,000 tons	1929	7	2	1	0	0	0	0	10	28.0	13.8
	1930	5	0	1	1	0	0	0	7	25.7	11.5
All mines	1929	28	19	11	0	0	12	2	72	-	-
	1930	34	5	11	2	1	8	0	61	-	-

Eleven fatalities occurred in 1930 and the same number in 1929 from contacts with electricity; two fatalities occurred in Class III - one from pushing a car under low, unguarded wire, and one from contact while a man was in a locomotive on the surface; seven of these fatalities from electrical contacts, all due to low, unguarded trolley wire, were in Class IV mines; the one in Class VI was a case of touching the trolley harp of an electric locomotive while replacing the wheel on the wire at a point where a guard had been placed on only one side of the trolley wire. Two mines in Class IV had three deaths each from electrical contacts. Trolley wire should be raised or guarded in some manner wherever it is less than $6\frac{1}{2}$ feet above the rail, or storage-battery locomotives should be used if deaths from electrical contact with trolley wire are to be eliminated.

Haulage fatalities dropped from 19 in 1929 to five in 1930. Two in Class IV mines were caused by falling from loaded trips; 3 occurred in Class V mines, 1 when a miner pushed a loaded car onto a main haulage road, 1 when a surface wreck occurred, and 1 while a trip was being pushed. The improvement in this type of accident is perhaps due chiefly to curtailed production, for there probably is less nervous haste and the equipment is better able to keep up with the work required when the tonnage requirements are lessened.

Two deaths were caused when men were run over in rooms by mining machines.

A poorly insulated blasting cable and improper requirements in shooting caused one fatality by explosives in a Class V mine.

An explosion or ignition of gas in a Class III mine caused by an electric arc at a working face due to lack of inspection resulted in the death of seven men. One explosion in a Class IV mine was caused by an open light in a slightly gassy mine with poor ventilation.

Table 7 classifies the fatalities by causes and occupation of men:

Table 7. - Fatalities in Alabama in 1930 classified by causes and occupation

Occupation	Rock fall	Haul- age	Elec- tricity	Mach- inery	Explo- sives	Gas explosion	Miscel- laneous	Total
Superintendent		1						1
Foreman	3		1			1		5
Machineman	1		1	1		1		4
Machine helper	4		1	1				6
Motorman	2		2					4
Coupler		2	2					4
Driver		1						1
Miner	19	1	3		1	1		25
Leader	2		1			4		7
Timberman	1							1
Conveyor						1		1
Utility	2							2
Total	34	5	11	2	1	8	-	61

Table 7 shows the occupations of men killed by every main cause in 1930, and indicates that machinemen were occupied in the most hazardous work; 10 were killed - 5 by falls, 2 each by electricity and by machines, and 1 by gas ignition. Locomotive crews were second in severity according to occupation as compared to the number employed; there were 9 fatalities under this classification--2 from falls, 3 from haulage, and 4 from electricity. Thirty-two miners and loaders were killed--21 by falls, 1 by haulage, 4 by electricity, 1 by explosives, and 5 by gas ignitions.

The fact that, of a total of 61 killed in all occupations in Alabama's coal mines in 1930, five of the victims were foremen appears to indicate that at least some foremen were not giving the proper thought to working safely.

CONCLUSIONS

Apparently, the fatal accidents of 1930 in the coal mines of Alabama were due in a large measure to the neglect of operating officials to work safely themselves or to assume full responsibility for the safe working of the men under them. By the example of those who have maintained operations relatively free even from lost-time accidents, and others who are making every effort to improve their record, it is evident that a large part of the accidents now occurring in Alabama's coal mines can and should be prevented.

The small-tonnage mines evidently need to exercise much more care looking to safety in operation; those mines with annual tonnage under 50,000 had but 5.5 per cent of the State's production in 1929, but had 20.9 per cent of its fatal accidents, and in 1930 these same mines had 5.1 per cent of the production with 9.8 per cent of the fatalities. Neglect of safety in small mines is especially bad in the properties producing less than 10,000 tons per year.

It is evident also that much additional precautionary effort should be expended on the prevention of accidents from contacts with electricity; with 11 such deaths out of 72 fatalities in 1929 and 11 out of 61 in 1930, there is no question as to the seriousness of the electrical hazard in Alabama coal mines. The mines producing 100,000 to 200,000 tons annually appear to need especial vigilance in the use of electricity, as these mines had 4 of 11 of Alabama's coal-mine electrical fatalities in 1929 and 7 of 11 in 1930. In other words, these mines producing about 20 per cent of Alabama's coal in the years 1929 and 1930 had 50 per cent of the fatalities in Alabama coal mines from electrical contacts for the two years.

It is significant also that the 61 fatalities of 1930 in Alabama coal mines all occurred in but 34 of the 223 mines in operation; hence, the 189 mines free of fatal accidents are forced in some measure to bear the burden, or at least the stigma, due to the poor record of these 34 mines.

An inspection of Table 7 classifying the 1930 fatalities in Alabama coal mines by causes and occupations shows that additional precautions must be taken in the use of machinery in the coal mines of this State if the fatality rate is to be reduced. With 10 machinemen and helpers and 8 motormen

and helpers killed in 1930, plus 1 man killed by a conveyor, 2 by machinery, and 5 men--a foreman, three miners, and a loader--killed by electricity (in addition to employees on machines and locomotives also killed by electricity), it is evident that at least 26 of the 61 fatalities in Alabama coal mines in 1930 were due to machinery in one form or other. In view of the rapid extension of the use of machinery of all kinds in mining, far more precautions should be taken in the installation and use of all types of underground machinery and appurtenances, or the death rate will increase rather than decrease.

That Alabama's coal-mining industry is making progress can be seen in the much better accident rate of Alabama coal mines in 1928, 1929, and 1930, than in the past; it is also made evident in the numerous safety awards recently given to mines, mining organizations, and individual persons in Alabama. The Hull mine of the DeBardeleben Coal Corporation won the 'Sentinels of Safety Trophy' for having the best safety record of the bituminous-coal mines of the United States entered in the National Safety Competition in 1929, and the western division of the DeBardeleben Coal Corporation was awarded a certificate of honor by the J. A. Holmes Safety Association in 1930 for having worked an average force of 893 men for over two years without a fatality or a permanent total disability accident. At its meeting in Washington, D. C., in March, 1931, the J. A. Holmes Safety Association gave one of its certificates of honor to W. B. Hillhouse, chief mine inspector of Alabama, one to the Alabama Mining Institute for excellent service in forwarding safety in the coal mines of Alabama, and one to the Hull mine of the DeBardeleben Coal Corporation for having operated through 1930 with only one lost-time accident; at the Hull mine 11 months and 20 days of the year were worked without a lost-time accident in producing 100,699 tons during 301,763 man-hours of exposure.

I. C. 6520 September, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

SAFETY EDUCATION AT IRON MINES OF THE LAKE SUPERIOR REGION

BY

F. S. Crawford

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY EDUCATION AT IRON MINES OF THE LAKE SUPERIOR REGION[1]

By F. S. Crawford[2]

Companies which have been engaged in safety work for a number of years are becoming increasingly aware of the fact that it is not enough to supply guards for various pieces of machinery and equipment and to supply safety inspection to see that rules are being carried out. Studies of many safety organizations have demonstrated that the accidents which could have been prevented by proper guards constitute only a small percentage of the total accidents, and amount to perhaps not more than 15 per cent.

Evidently, if success is to be attained in preventing all accidents which it is humanly possible to prevent, a company must have the cooperation of every employee. Mining particularly presents probably the greatest hazards of any industry. Prevention of accidents is made increasingly difficult because men often work alone, although generally in pairs, and only in the concentrated systems of mining do they usually work in groups of more than three or four. Supervision is difficult, therefore, and much reliance must be placed upon the individual workmen. In mining, as perhaps in no other occupation, it is of the greatest importance that individual workmen should receive continuous safety education.

Within the last few years safety education has come to the front as one of the most important of accident-prevention measures, although it is true that its importance has been repeatedly stressed by the United States Bureau of Mines and other national safety organizations since the inception of the safety movement many years ago.

A review of the safety educational methods of some companies in the Lake Superior district should be of interest. Perhaps better methods are in use in other parts of the country, but the methods shown have been instrumental in many cases in reducing the number of accidents. In one mine in the district whose methods are described the frequency of accidents has been reduced to zero within the last two years. A number of other mines in the district have recently gone into the class having no lost-time accidents for considerable periods of time. One company in the Lake Superior region, operating over 30 mines, succeeded during one month of 1930 in operating every mine without having a single accident which caused lost time beyond the day of the injury. Records like these are increasing in number, and while it is believed that they are the combined result of different means of safety activity, it is undoubtedly true that many of these results were obtained through safety educational methods.

COMPANY 1

Bulletin-board service is maintained at every property of Company 1. The principal board is at a point where it can be most easily observed by the largest number of employees.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6520."

2 - District engineer, United States Bureau of Mines Safety Station, Duluth, Minn.

Other bulletin boards are maintained at outlying points of the mine. The display on the boards is changed every two weeks.

First-aid training is conducted twice a year at the general foremen's meetings to keep the foremen familiar with this work. A number of the employees have taken the mine rescue training offered by the United States Bureau of Mines, but no mine-rescue apparatus are maintained by the company.

COMPANY 2

At company 2 bulletin-board service is maintained through subscription to a commercial service. These boards are set up at strategic points at all operations, and posters of world events and safety posters are changed every week. No educational classes have been held at this property, and up until 1930 there had been no first-aid training. In 1930, however, about 40 men were trained by the United States Bureau of Mines. No attempt was made to give the mines 100 per cent first-aid training because operations, including underground mines, had been shut down due to lack of orders and approach of the cold weather. The number of men trained in first aid comprised about 80 per cent of the working force that had been retained for last winter's work; these men showed such interest that the company decided to train the remainder when the mines reopend last spring. This company is entering more actively into safety work.

Foremanship training is done to some extent through the semi-monthly foreman's bulletin which is received from the commercial bulletin-board service.

COMPANY 3

Safety education at company 3 is confined to bulletin-board service and "movies" furnished by an insurance company. Bulletins of the commercial bulletin-board service and the insurance company service are placed upon the board.

No educational classes have been held at night on the part of the company, except the first-aid classes under the direction of the United States Bureau of Mines, in which all the plants were given 100 per cent first-aid training.

COMPANY 4

Company 4 is small, with three shafts serving interconnected workings. Bulletin boards are furnished and are posted with bulletins supplied by a commercial service and by the National Safety Council. No first-aid classes were held by the company until safety meetings were started a short time ago, but now training in first aid has been given to 100 per cent of the men by the United States Bureau of Mines.

The company contributes to a cooperative safety station maintained by one of the larger companies in the district. There have been no established foremanship classes, all instruction of the foremen having been given by personal contact. The mining engineer having charge of the bulletin-board service passes on the bulletins received from this service to the public school in the village.

COMPANY 5

Bulletin boards are placed in the various plants of company 5 and are posted daily with the bulletins of a commercial service, the National Safety Council, and an insurance company. In addition to these bulletins there are also homemade bulletins on safety, sports, monthly accident reports, and other subjects.

Classes in citizenship are given every year in the public schools. First-aid classes are held once a month at each mine. First-aid contests are held at irregular intervals.

Mine rescue training is given once a month to about 50 men in each of the districts. This company has about 70 sets of self-contained breathing apparatus available in Michigan and Minnesota. Foremanship classes are held once a month at the monthly safety meetings, supplemented by the foremanship service of the commercial bulletin-board service.

Safety education is carried on to some extent in all public schools in the district, those schools doing such work are members of the National Safety Council whose calendars are furnished to each public school room by the mining company.

COMPANY 6

The bulletin boards of a commercial service are used at all properties of company 6. In addition to this service there are also displayed National Safety Council bulletins, local posters, honor rolls, announcements, and newspaper clippings of accidents at other companies' mines. When a man breaks his goggles with the saving of his eyes, the broken glasses are displayed as a warning to all men to wear their goggles.

The company has eight self-contained breathing apparatus and six All-Service gas masks available in case of mine fires. Forty men have been trained in the use of mine rescue breathing apparatus. These men are retrained every three months by the company safety engineer.

Some safety education is done in the public schools through the cooperation of the company. There is a safety talk at community meetings, followed by safety movies and other educational films and a two-reel comedy. A five-piece orchestra is generally furnished to give music, and after the meeting there is dancing and a lunch.

Twenty community meetings were held in 1929 at the various properties of this company as follows:

Mine	Meetings	Attendance
1	4	1250
2	3	850
3	2	500
4	2	400
5	2	400
6	1	400
7	2	375
8	2	300
9	1	175
10	1	175
Total	20	4825

COMPANY 7

Bulletin boards are placed at every mine of company 7. Upon them are placed safety posters supplied by the National Safety Council, comparative accident records by graphs and statistics, and descriptions of preventable accidents. In addition, there is a regular monthly safety bulletin discussing the accidents of the previous month, as well as civic news items of interest which will attract the men to the board.

Night classes for foremen were held weekly for two years. After the completion of this course it was not continued. Meetings of mechanics and electricians were held monthly during the fall and winter months.

Of a total of 2,112 men employed by the company in 1929, 739 were first-aid trained. Teams of five men each are trained every month at each mine. As a rule an exhibition first-aid contest is held every year, in which only the employees of the company take part.

The company now has 422 employees who have been given mine rescue training. Select groups of men are given additional training at each of the underground mines every month. The company has available 25 self-contained breathing apparatus and has several mine rescue stations equipped with all the necessary supplies and equipment for servicing the machines and for fire fighting.

Conferences of the foremen are held about four times per year in each district, and one or two general conferences are held annually. The company does not carry on any safety education in the schools of its district, although some of the schools are members of the National Safety Council and are carrying on their own safety work.

COMPANY 8

Company 8 maintains bulletin boards at each plant. The displays on these boards are changed every day and usually contain a poster on general or mining safety subjects published by the National Safety Council or by some other industrial poster service. In addition, there are posters made by the safety engineer on various subjects which he wishes to bring before the men, and rotogravure news bulletins which serve to attract the men to the board.

First-aid classes are held periodically, depending upon the number of new men hired. It has not been the custom in recent years to hold interplant first-aid contests. These were last held in 1916, and first-aid demonstrations were given in 1925. Prior to 1930, about 40 per cent of the men had been trained in first aid. During 1930, with the cooperation of the United States Bureau of Mines, 100 per cent first-aid training was given at all the plants of the company. There are 24 men who have United States Bureau of Mines certificates of proficiency in mine rescue work. Additional men are trained periodically by the United States Bureau of Mines personnel. Twenty men have also been given the advanced course in rescue training. Ten men inspect the rescue apparatus monthly, and are drilled five to six times a year. Of the underground properties operating, one of them is equipped with five two-hour McCaa oxygen breathing apparatus and the other with five Burrell All-Service gas masks, supplemented with other necessary equipment for rescue work and fire fighting.

The company has not taken up foremanship classes, nor has it done anything in connection with safety education in the public schools. The schools on the Mesabi range of Minnesota, where the company operates open-pit mines, have some safety education, several of them being members of the National Safety Council. These issue safety and accident statistics monthly. While making trips about the mines, various officials give instructions to the men. These officials include the superintendents, the mining captains, shift bosses, safety inspectors, and safety engineers.

COMPANY 9

Company 9 subscribes to a commercial bulletin-board service. The board is made up of three panels. The center one is used for pictures of current events and world news, and the side panels are used for safety posters and pictures on safety. The pictures and news notes are changed daily.

No night classes have been held for the purpose of safety training or foremanship. Approximately 130 men were given first-aid training by the United States Bureau of Mines instructor when 100 per cent first-aid training was carried on at one of the mines. All first-aid and mine rescue men were given special training in artificial respiration.

Twenty-four men have been trained in the use of mine rescue oxygen breathing apparatus. Five sets of self-contained breathing apparatus are on hand, together with two All-Service gas masks and four canisters, two stretchers, four blankets, and a complete assortment of splints. The company gives additional training to the men at intervals of several months as well as training new men in use of apparatus.

Education in foremanship is carried on through the books on foremanship issued by the commercial service to the company twice a month. The schools and local civic organizations also carry on safety activities.

COMPANY 10

Company 10 places bulletin boards in the surface shops, in the "dry" or change house, and at underground shaft stations of the mine. On them are placed bulletins of the National Safety Council, as well as the posters of a commercial service and local posters formulated by members of the safety organization.

The company has held no regular night classes. However, first-aid classes which have formerly been held in the daytime on company time are now held on the men's own time, and some of these are given at night, about once a year. The United States Bureau of Mines trained all of the men at this operation in first aid during 1930, and the company plans to train new men from time to time.

Forty men have been trained in the use of mine rescue breathing apparatus. The company has five self-contained breathing apparatus and ten All-Service gas masks. The men are retrained about once in two months by one of the assistant safety engineers as well as by the personnel of the United States Bureau of Mines when they are available in the district. The company does not hold any classes in foremanship, but subscribes to the foremanship pamphlets of a commercial service, which are distributed to all foremen.

While there is no organized safety education carried on in the public schools with the cooperation of the company, the safety engineers give occasional talks on safety before the school children.

The safety inspectors conduct a more or less continuous educational campaign on the safety rules of the company. This is done by reading the rules to the underground men while on shift in their working places. They explain the rules and clear up any misunderstanding which the men may have regarding them.

COMPANY 11

Company 11 displays bulletin boards in the shops, change house, and at the various underground levels; they are all equipped with electric light and placed where they will best attract the attention of the workmen. The company uses the posters of the National Safety Council and also displays safety material made at its own property. These bulletins are changed every week.

It has been customary to give weekly training in first aid to a small class of men. During 1930 the company had 100 per cent of its men trained in first aid by the United States Bureau of Mines, and future plans are to train all new men through company instructors and submit these men for examination by the bureau. No first-aid contests are held.

A small class is trained in the use of mine rescue apparatus once a month, following the instructions as set forth in the United States Bureau of Mines manual on mine rescue training. The company keeps four teams of mine rescue men trained -- a total of 20 men -- giving each team one practice period a month.

The rescue equipment includes five self-contained two-hour breathing apparatus, four All-Service gas masks, seven self-rescuers, and enough supplies on hand to last 36 hours. They can also call upon the other properties of the sales agent which are equipped with the same amount of apparatus.

The company does not hold any foremanship classes. Safety is taught in every grade in the schools in the township.

Attached is a summary of the safety program as taught in every grade. The schools are also members of the National Safety Council. The safety engineer gives safety talks in the public schools.

Safety cards are placed in the time-card racks in the change house, a copy being given to each employee. On these cards are printed safety rules and methods of doing work safely. A new card is distributed every month. Following is a sample card:

EMPLOYEES SAFETY CARD

Card No. 15.

(Read this card carefully before going to work.)

General Rules

Before going on a platform or scaffold of any sort, see that it is properly built.

Tools and timber must not be thrown down manways. When they are being hoisted or lowered, they must be securely lashed, or other suitable device used. Care must be taken to see that no one below is in danger.

No one shall take timber or ladders which are in place unless directed to do so by the foreman.

Be careful of falls from the back when taking down your drill column. The back may have been jarred loose by drilling.

Goggles must be used wherever there is danger of injury to the eye. Specific instances are: Starting holes, drilling uppers, picking loose, breaking chunks, standing near anyone breaking chunks, using a blow pipe, chopping back lagging, chopping timber above the waist line, cutting wire cables. In cases where other hazards may outweigh the eye hazard the foremen may instruct their men not to use goggles.

Gloves must be worn wherever there is danger of injuring your hands.

Everybody

Remember that all the rules and regulations that can be adopted all the safety devices that can be attached to machines, all the guards that can be erected, all the warning signs that can be posted are useless unless every man is careful to see that they are maintained, and unless every man is careful to warn others of danger.

OUTLINE OF COURSE OF STUDY IN SAFETY AT BESSEMER TOWNSHIP SCHOOLS, GOGEBIC IRON RANGE, MICHIGAN

The following is an outline of the safety activities that are taught in the Bessemer Township schools: namely, the Ramsay, Anvil, Puritan, and Harding Public Schools.

OUTLINE OF COURSE OF STUDY IN SAFETY AND FIRST AID

Grade II

The importance of safety habits with reference to:

1. Railroad crossings - street crossings.
2. Hanging wires.
3. Guns.
4. Skins of fruit.
5. Matches.
6. Rusty nails.
7. Animals.
8. Slippery places.
9. Stairs.

Grade III

Dangers to guard against:

1. Spitting on floor or sidewalk.
2. Roller towel - public towel.
3. Refuse.
4. Flies.
5. Odors.
6. Teachers and pupils extend list.

Safety rules and habits - one each month by seasons:

1. At crossings.
2. Hanging wires.
3. Firearms.
4. Fruit skins and cores on stairs or street - prevent falls.
5. Matches and fire.
6. Rusty nails.
7. How to get on and off bus and street car.
8. Where to play safely.

Grade IV

Safety - List dangers from accidents due to:

1. Automobiles.
2. Street dangers - coasting.

Grade IV - Continued

3. Fires - how to tell where located and give alarm.
4. Poisons - how to recognize.
5. Electrical accidents.
6. Falls and falling objects.
7. Railway tracks and trains.
8. Weapons and firearms.
9. Contagious diseases.

First aid - What to do in case of accident:

1. Call an older person - how to telephone.
2. Call doctor if accident is serious.
3. How to stop bleeding.
4. What to do if clothing is on fire.
5. How to remove objects from the eye:
   a. Do not rub eye.
   b. Hold lid down so tears will wash it out.
   c. Go to an older person.
6. Nose bleed:
   a. Hold head back.
   b. Cold cloths on nose and back of neck.
7. Frosted hands, feet or face:
   a. Rub with snow or cold water.
8. Open wounds should be kept clean.

Grade V

First aid - What to do in emergencies:

1. What to do in case of fire.
2. What to do in case of broken arm or leg.
3. What to do for a nose bleed.
4. Care of cuts, bruises, and burns.
5. Use of iodine.
6. How to stop bleeding in various parts of the body:
   a. Make charts
7. How to prevent colds:
   a. By keeping the body fit.
   b. By proper diet and rest.
   c. By isolation of affected persons.
   d. Use of handerchief when coughing or sneezing.
8. What to do when exposed to contagion.

Safety:

Note: There are ten topics in the fifth grade. In general, one topic a month should be studied.

Grade V - Continued

1. Street Dangers:
   a. Talk with children about street dangers and make a list of all the dangers of the streets. Why are these dangers so much greater than they used to be?
   b. Make slogans and jingles on crossing the street as:
      "He who learns to look each way,
      Will live to look another day."
   c. Make a newspaper collection of street accidents and keep for reference.
   d. Call attention to dangers of caves and abandoned shafts in the vicinity. These should be fenced. They are especially dangerous when snow covers the ground.
   e. Danger of contagion.
2. Fires:

   There is a fire somewhere in the United States every minute of the year. Nearly a half billion dollars is lost and 16,000 die from burns each year. More than this number are injured.
   a. Observe FIRE PREVENTION WEEK. Make use of the daily papers for materials.
   b. Jingles:
      "Jack be nimble, Jack be quick,
      But dont knock over the candlestick."
   c. Collect material on fires from fire insurance companies.
   d. What to do in case of fire.

3. Weapons:

   These include all kinds of weapons, guns, air rifles, sling shots, bows and arrows, sharp sticks, clubs, stones, knives, etc.

   a. Write a story about "He didnt know it was loaded."
   b. Slogans and jingles.
   c. Make a collection of newspaper accounts of children and persons injured by various weapons.
   d. What are the dangers from Fourth of July celebrations?
   e. Debate the benefits and dangers from air guns.
   f. Make a collection of hunting accidents reported in the Daily Globe for Upper Peninsula.

4. Burns and Scalds:

   Because fire has a natural attraction for children, this topic will evoke much interest on the part of the children.

   a. Make a list of the ways by which people are accidentally burned.
   b. Make a set of rules to prevent accidental burning.
   c. Make a list of old slogans, like - "A burnt child dreads the fire."
   d. Let children prepare a play on safety in the kitchen. Use an appropriate title.

Grade V - Continued

5. Poisons and Asphyxiation:

   a. Introduce the study of this section by listing and talking about all the sources of poisoning that may be around the home.
   b. What should one do if he smells gas in the home?
      "Jack be nimble, Jack be quick,
      But dont look for gas with a candle stick."
   c. If one is found unconscious by gas poisoning, the prone-pressure method of resuscitation should be used as long as any signs of life remain. Demonstrate the prone method of resuscitation until children are familiar with it.
   d. Draw pictures of poison labels, poison plants, etc.

6. Electrical Safety:

   a. The great importance and wide use of electricity:
      1. A source of heat, light and power.
      2. How it is carried from one place to another:
         (a) Good and poor conductor.
         (b) The human body as a conductor.
         (c) Effect on the human body.
      3. Why "fuses" are used in the home.
   b. Compile a list of the requirements of city ordinances concerning the wiring of buildings. Why are those laws necessary?
   c. What are fuse plugs? Why necessary? Is electrical safety valve a good name for a fuse plug?
   d. What should children do if they find a broken electric wire on the ground?
   e. Why do electric lights often go out during a thunder storm? A wind storm? A rain storm? Dangers of a fallen "live wire."

7. Falls and Falling Objects:

   The cold winters and prevalence of much ice on the sidewalks and streets make this a topic that should receive careful attention. The most dangerous places on the sidewalks are spots that have been worn slick by children skating or sliding. A bit of snow often falls on these slick places and thus a person may venture on such dangerous places without thought of harm.

   What can children do about it? Would it be a good turn for a Boy Scout to use his axe or knife to roughen the surface of such slick places?

   a. Climbing. Discuss dangers. Reasons why children often fall (Bodily control not yet established) Where do children climb?
   b. Make a list of all falling objects that might be dangerous. Are there any laws which require people to be careful how objects are placed or supported? Who would be liable in case a hammer fell from a building under construction and injured a man passing on the sidewalk?

Grade V - Continued

c. In the home, most accidental falling comes from poorly constructed stairways. What kind of stairways are most dangerous? What kind are safest? Perhaps some child can make a clay model of one or both kinds.
d. Project: Divide your room into those who have been hurt by falls and those who have not. Then make a list of the causes and find which is most common and how many falls could have been prevented.

8. Railway tracks and trains:
   a. Make up other rhymes and jingles:
      "Keep off the tracks, avoid disaster,
      Trains are a million times bigger and faster."
   b. Collect and exhibit a group of pictures to show the dangers of trespassing on railroads and in crossing railroads.
   c. Outline the work that a Junior Safety Patrol might do in preventing trespassing upon the railroad tracks.

9. Accidents in play:

   a. Make a list of dangerous kinds of play and places of play in your locality.
   b. Prepare a list of the kinds of accidents that you know have happened to children.
   c. Draw pictures and make posters showing how children get hurt at play in the streets.

10. Drowning:

    Play activities which may result in drowning are so fine a form of exercise that, like football, it is felt more would be lost than gained by abandoning them. Bathing, swimming, diving, skating, and boating are so exhilarating and attractive that no one would think of being away from them. It is possible, however, through care and knowledge of what to do in emergencies, to reduce hazards to a minimum.

    a. Causes of drowning:
       1. Deep water.
       2. Very cold water.
       3. Cramps of the muscles.
       4. Fall from a boat or pier.
       5. Breaking through ice.
       6. Stepping into a deep hole when wading.
       7. Diving into unknown depth.
       8. Undertow.
       9. Overexertion.
    b. Make a set of rules which, if followed, would make swimming and boating reasonably safe.
    c. What to do in case of possible drowning:

Grade V - Continued

1. Go for help or send some one.
2. Save victim if you can.

d. Demonstrate the prone pressure method of resuscitation. Teachers should be sure about the correct procedure in this method. See Boy Scout Handbook or Red Cross Bulletin 1004: Junior Life-saving Crews.

Grade VI

First Aid:

1. Teach what to do, and have pupils actually perform first aid methods in following cases:
   a. Bleeding - how to stop.
   b. Burns.
   c. Cuts - bruises.
   d. Drowning - resuscitation.
   e. Other suitable activities.

Safety:

Note: There are ten topics in the sixth grade. In general, one topic a month should be studied.

1. Street Dangers:

   a. Review fifth-grade work.
   b. Write a playlet or put on an assembly program.
   c. Make a list of safety rules and save the best ones for a safety exhibit.

2. Fires:

   a. Review fifth-grade work.
   b. Make a simple graph of the loss in life by fires for the past five years. Do the same for property loss.
   c. Why is parking near a fire hydrant prohibited? What must vehicles do when a fire engine is racing to a fire?
   d. Make a list of ways to prevent fires.
   c. Does the fact that a building is insured mean there is no loss?

3. Weapons (see Grade V):

   a. Procure figures on the number of people killed and injured yearly by weapons.
   b. What laws govern the sale and use of firearms?
   c. Make safety rules for the use of firearms.
   d. Why do so many accidents occur in the woods here during hunting season?
   e. Project: How to celebrate the "Fourth" in a safe way and yet have fun.

4. Burns and Scalds (see Grade V):

   a. Project: Show home dangers from burns and scalds by means of toy utensils.
   b. Learn or write a play "Who's to Blame" or "The Match and the House." Put on for school assembly if good enough.
   c. Why is cleaning by gasoline, benzine, or naphtha so dangerous.

5. Poisons and Asphyxiations:

   a. Teach fully the meaning of poisoning and asphyxiation.
   b. Unconsciousness from electric shock is often due to asphyxiation or a paralysis of the muscles used in breathing. One unconscious from such cause should be revived by the prone method of resuscitation--the same as in the case of drowning. If the injured one is still in contact with the electric wire causing the shock, do not touch the body, but use a board or stick and first free the person from the contact.
   c. Demonstrate fully the method of reviving people who have ceased to breathe, or have been asphyxiated.
   d. Find out if there is any danger from asphyxiation in the mines. How is asphyxiation prevented? What poisonous gases are encountered in the mines?
   e. Prepare a list of antidotes for the most common kinds of poisoning. What should be done in the event that some one drinks a poison by mistake?
   f. Debate: "Resolved: That the person who spreads disease by neglecting quarantine is as guilty as one who places poison where children may get it."

6. Electrical Safety (see Grade V):

   a. Two kinds of danger from electricity:
      1. From fire caused by heating of wires.
      2. From a flash or "arc" when current flows from one conductor to another.
   b. Study cases of deaths and injuries by electricity; analyze by causes and show how death could have been prevented.

7. Falls and Falling Objects (see Grade V):

   a. Make a list of dangers from falling objects in your locality.
   b. Dramatization: Plan a trial of causes of falls. Let a child or dummy be placarded with a given cause and tried. Have a prosecutor and a defending lawyer.

8. Railway Tracks and Trains (see Grade V):

   a. Oral accounts of notable railway experiences of home folks.
   b. Locate the most dangerous crossing in and about your locality. What care and caution should be exercised at each place?

c. Write to the central department of safety of the Chicago & North Western Railway Co. at Chicago for printed matter and helpful suggestions.

9. Accidents in Play (see Grade V):

a. Make up slogans and rhymes concerning safety in play.
b. Make up posters in art classes on safety.
c. Write a playlet or dramatize a story or play for an assembly program. Use a theme such as the boy who coasted on the street, or hitched onto vehicles in spite of the warnings of his parents and friends.
d. Prepare a list of dangerous games which children play in your locality.
e. What laws should govern the playing of children in the streets?

10. Drowning (see Grade V):

Questions from Safety Education, Chicago Board of Education:

a. What is drowning?
b. How do people drown in a bathtub?
c. What dangerous games do boys play in swimming?
d. What is the danger of standing and rocking in a small boat?
e. Give the history of some of the noted drowning accidents.
f. Make a story of any family experience relating to drowning.
g. Make a story of any personal experience relating to drowning.

Make sure that all children understand and can perform the prone-pressure method of resuscitation. (See Grade V for reference).

Review the safety work for the year in assembly program, including the work of the fifth and sixth grades.

I. C. 6521

October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SAFETY IN THE IRON MINES OF THE MENOMINEE RANGE, MICHIGAN

BY

F. S. Crawford

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY IN THE IRON MINES OF THE MENOMINEE RANGE - MICHIGAN[1]

By F. S. Crawford[2]

After many years of safety activity in the United States in one form or another it becomes increasingly evident to those in contact with safety work that to achieve continuous good results in accident prevention there must be an aggressive organization behind the safety movement at each mine with fairly definite plan and policy. A study of the methods used by various companies and the results they have attained will be of value to the common cause of safety promotion.

A study has been made of the safety records of the Menominee iron range in the Lake Superior district, and of data from the county mine inspector's report. Several companies on this iron range that have attained a certain measure of success in their work of accident prevention have given data, and a summary of their safety methods is presented.

RECORD OF MENOMINEE RANGE

The Menominee iron range of Michigan lies in the southern portion of the upper peninsula of Michigan, bordering on Wisconsin. It consists of two main divisions--the West Menominee Range, situated in Iron County in the vicinity of Iron River and Crystal Falls, and the East Menominee range situated in Dickinson County in the neighborhood or Iron Mountain, Mich. The mines about Iron River and Crystal Falls are peculiarly subject to fires in a carbonaceous, sulphur-bearing black slate which appears in the footwall. This gives considerable trouble because fires in this material start from spontaneous heating. The methods of mining are top slicing and open stoping, working on benches along the sides of the stopes. The character of the ground varies from medium soft ore to relatively hard ore which permits open stoping. Approximately one-third of the operating mines are equipped with a fan for main mechanical ventilation. The remainder are ventilated by natural draft. The East Menominee range is the older, top slicing having been used there for a number of years, and the timber mats have been well developed. In common with all districts, some companies pay more attention to safety than others, although those which do not have the best records do pay some attention to safety.

1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6521."

2 District engineer, United States Bureau of Mines Safety Station, Duluth, Minn.

During the past three years the annual production per man employed has remained about constant, but in many mines a general decrease is shown in the accident record. The variation in the results attained can be ascribed mainly to the earnestness with which accident-prevention work is pursued, essentially regardless of the method of mining used. This is shown to be generally true, because the record indicates a steady decrease in one mine with an active safety department, while in adjoining mines with the same kind of ground and method of mining but where not such intensive attention has been paid to safety work, the accident record is high.

The county mine inspectors' reports for Dickinson and Iron County show the following injury record for the Menominee Range (Table 1):

Table 1.-Injury record for the Menominee range, 1924-30.

IRON COUNTY	1924	1925	1926	1927	1928	1929	1930
Total fatal and lost-time accidents per 100 men employed	16.5	16.5	15.6	14.0	8.9	6.5	4.5
Total minor accidents causing 1 to 10 days lost time per 100 men employed	6.3	6.8	6.0	4.4	3.1	3.3	2.4
Total fatal and serious accidents causing over 10 days lost time per 100 men employed	10.2	9.7	9.6	9.6	5.8	3.2	2.1
DICKINSON COUNTY							
Fatal and compensable[1] accidents per 100 men employed	1.7	2.4	2.4	1.6	2.2	1.6	2.4

1 Compensable accidents only are reported. This includes those causing more than 7 days lost time beyond the day of injury.

The two counties do not report their accidents in the same way, so it is impossible to make a summary for the Menominee range as a whole on a common basis. The Iron County report shows the lost-time accidents divided into those causing time lost of 1 to 10 days, 10 to 30 days, and 30 days or more. The 1 to 10 day classification includes all the noncompensable lost-time accidents and some compensable accidents which have gone beyond seven days. However, the foregoing record shows that Dickinson County has enjoyed a low injury record for the period from 1924 to 1930, with an average for the period of approximately two compensable or fatal accidents for each 100 men employed. We may consider a fatal injury as a serious injury which has resulted fatally; some serious injuries miss being fatal only by a very narrow margin. Two of every 100 men are seriously or fatally injured in the Dickinson County mines every year; this may be interpreted to mean that on the average a man may work in Dickinson County mines 50 years before he is seriously injured.

The record for Iron County does not start out well in 1924, but much progress has been made since that time; the accident rate has dropped steadily. Some companies which had been carrying on more or less safety work have attacked the problem with greater wigor and with some change of method in the last few years. The minor accidents causing time lost from 1 to 10 days have been steadily reduced from 6.3 per 100 men in 1924 to 2.4 in 1930. The number of those more seriously and fatally injured has also dropped steadily from 10.2 in 1924 to 2.1 in 1930. A continuation of the present rate will make the chances of avoiding injury in Iron County mines practically equal to those of Dickinson County. Thus, the average miner on the Menominee range, based on the record so far attained in both counties, may work 50 years before being seriously or fatally injured.

There are nine companies producing iron ore in Iron County and operating 24 mines. In Dickinson County, companies which also operate in Iron County operate six mines, producing practically all of the output. Tables 2 and 3 list the individual records of these companies.

Table 2.- Accident record of Menominee iron range in Iron County

Item	Company									Total
	A	B	C	D	E	F	G	H	I	
1924										
Average men employed	795	776	305	140	203	222	67	70	86	2820
Fatal accidents	1	5	0	0	0	1	0	1	1	9
Lost-time and fatal accidents	70	147	65	20	19	30	6	22	12	446
Fatal accidents per 1,000 employed	1.3	6.5	0	0	0	4.5	0	1.4	1.2	3.2
Lost-time and fatal accidents per 100 employed	8.8	18.9	21.3	14.3	9.4	13.5	8.9	31.4	13.9	16.5
1925										
Average men employed	883	914	293	130	190	273	78	65	34	2695
Fatal accidents	1	2	1	0	1	3	0	0	0	8
Lost-time and fatal accidents	140	152	61	18	16	49	13	15	2	444
Fatal accidents per 1,000 employed	1.1	2.2	3.4	0	5.3	11.0	0	0	0	3.0
Lost-time and fatal accidents per 100 employed	15.8	16.6	20.8	13.8	8.4	18.0	16.7	23.0	5.9	16.5
1926										
Average men employed	745	806	333	100	175	267	105	71	49	2655
Fatal accidents	0	1	0	0	0	0	0	1	0	2
Lost-time and fatal accidents	108	138	53	17	6	45	16	16	5	415
Fatal accidents per 1,000 employed	0	1.2	0	0	0	0	0	1.4	0	2.6
Lost-time and fatal accidents per 100 employed	14.5	17.1	15.9	17.0	3.4	16.9	15.2	22.6	10.2	15.6
1927										
Average men employed	735	757	356	110	175	175	124	67	55	2611
Fatal accidents	2	2	0	0	0	1	1	1	0	7
Lost-time and fatal accidents	138	123	5	27	3	29	13	18	9	366
Fatal accidents per 1,000 employed	2.7	2.6	0	0	0	5.7	8.1	1.5	0	2.8
Lost-time and fatal accidents per 100 employed	18.7	16.2	1.4	24.6	1.7	16.6	10.5	26.8	16.4	14.0

Table 2.- Accident record of Menominee iron range in Iron County - Continued

1928

Item	A	B	C	D	E	F	G	H	I	Total
	Company									
Average men employed	700	647	380	113	187	159	100	63	57	2452
Fatal accidents	2	2	0	1	0	0	0	1	0	6
Lost-time and fatal accidents	114	36	3	20	5	18	3	11	5	219
Fatal accidents per 1,000 employed	2.9	3.1	0	8.9	0	0	0	15.9	0	2.5
Lost-time and fatal accidents per 100 employed	16.3	5.6	0.8	17.7	2.7	11.3	3.0	17.4	8.8	8.9
1929										
Average men employed	618	516	407	147	182	108	88	58	61	2287
Fatal accidents	3	3	2	0	0	1	0	2	0	11
Lost-time and fatal accidents	58	14	7	33	7	12	4	9	4	148
Fatal accidents per 1,000 employed	4.9	5.8	4.8	0	0	9.3	0	34.5	0	4.8
Lost-time and fatal accidents per 100 employed	9.4	2.7	1.7	22.4	3.8	11.1	4.5	15.5	6.6	6.5
1930										
Average men employed	613	521	405	164	146	237	84	60	39	2265
Fatal accidents	2	5	0	0	0	1	0	0	0	8
Lost-time and fatal accidents	17	12	5	25	9	22	3	4	4	101
Fatal accidents per 1,000 employed	3.3	9.6	0	0	0	4.2	0	0	0	3.5
Lost-time and fatal accidents per 100 employed	2.8	2.3	1.2	15.3	6.2	9.3	3.6	6.7	10.0	4.5

Table 3.- Accident record of Menominee iron range in Dickinson County

Item	Company A	Company B	Company E	Total
1924				
Average men employed	347	80	944	1371
Fatal accidents	0	0	0	0
Lost-time and fatal accidents	12	2	8	22
Fatal accidents per 1000 employed	0	0	0	0
Lost-time and fatal accidents per 100 employed	3.5	2.5	0.8	1.7
1925				
Average men employed	377	76	805	1258
Fatal accidents	1	0	5	6
Lost-time and fatal accidents	6	10	14	30
Fatal accidents per 1,000 employed	2.7	0	6.2	4.8
Lost-time and fatal accidents per 100 employed	1.6	13.2	1.7	2.4
1926				
Average men employed	391	73	777	1241
Fatal accidents	1	0	2	3
Lost-time and fatal accidents	9	8	13	30
Fatal accidents per 1,000 employed	2.6	0	2.6	2.4
Lost-time and fatal accidents per 100 employed	2.3	11.0	1.7	2.4
1927				
Average men employed	381	103	803	1287
Fatal accidents	0	0	2	2
Lost-time and fatal accidents	2	3	15	20
Fatal accidents per 1,000 employed	0	0	2.5	1.6
Lost-time and fatal accidents per 100 employed	0.5	2.9	1.9	1.6
1928				
Average men employed	355	137	764	1256
Fatal accidents	0	0	1	1
Lost-time and fatal accidents	1	8	19	28
Fatal accidents per 1000 employed	0	0	1.3	0.8
Lost-time and fatal accidents per 100 employed	0.3	5.8	2.5	2.2
1929				
Average men employed	340	141	732	1213
Fatal accidents	1	0	3	4
Lost-time and fatal accidents	8	2	9	19
Fatal accidents per 1,000 employed	2.9	0	4.1	3.3
Lost-time and fatal accidents per 100 employed	2.4	1.4	1.2	1.6
1930				
Average men employed	332	145	697	1174
Fatal accidents	1	0	2	3
Lost-time and fatal accidents	5	5	18	28
Fatal accidents per 1,000 employed	3.0	0	2.9	2.6
Lost-time and fatal accidents per 100 employed	1.5	3.5	2.6	2.4

Three companies employing 1,539 of the total of 2,265 men in Iron County mines had only 34 of the total of 101 time-lost and fatal injuries of all descriptions. The other companies, employing the remaining 726 men, had 67 accidental injuries. The rate for the first group was 2.2 per 100, whereas the rate for the second group was 9.2 per 100. The high accident rates are found in the mines of small companies which, however earnest is their desire to prevent accidents from either a humanitarian or an economic consideration, have not yet succeeded in using the proper means to accomplish this purpose. It is apparently more difficult to organize safety activities at the single small operation than at the single large mine or group of mines, large or small, of the large company (this is so in other industries); much of this difficulty is probably due to the fact that the small operator feels that he can not afford the services of a safety engineer and rarely has anything like an effective safety organization or working plan.

There is now no reason why a fairly large company, with several mines, should continue to have a bad safety record; this also means that the smaller companies should be able to secure similar results by combining their efforts, at least in the forwarding of safety work.

By organization of safety committees to inquire into the causes of accidents, engaging in educational work among the men, and using some measure of discipline to back up the safety rules, the men can be brought to be more and more safety conscious and thus avoid accidents and resultant personal injuries. The employment of a safety inspector, engineer, or director is generally necessary to see that the work is carried on steadily. The small company usually feels that it can not afford the added expense of the safety engineer; in many cases the question might well be asked, Can the operation afford not to have a safety engineer, and can it afford not to carry on accident-prevention work? Numerous articles have appeared recently in the technical press and in the proceedings of various engineering and safety organizations showing the direct and indirect causes of accidents. Some small mines may operate well without a safety engineer, but someone should be responsible for carrying on a ceaseless safety program, in more or less detail. In a small operation this may be made part of the duties of the mining engineer or assistant superintendent, but the higher up among the official personnel is the position of the man in charge of safety work, in general, the better will the work succeed. The small operator might well look upon the safety man as primarily the safety director, who would also be assistant to the superintendent or assistant superintendent as well, and employ his time with efficiency and operating problems when not engaged in safety activities.

The fundamental requirement for the success of accident-prevention work at any operation is that the management should be heart and soul and aggressively behind the safety work; following this, the men should be made to realize this attitude of the management. Besides continual emphasis on wise safety regulations, all rules should be backed up by discipline.

Of the nine companies operating 30 mines on the Menominee range, all but two small companies have a safety engineer. One of the small companies recently established a safety organization in charge of a safety engineer, and this action is showing good results, as for as may be judged by the record of one year; the injury record for 1930 is less than half that of 1929. The two companies which do not have a safety engineer are not showing consistent results in preventing accidents, although their foremen and superintendents are quite earnest in their desire to do so. The injury rate is low one year and high the next, and in general is considerably higher than at the other mines which have active safety organizations. The safety practices of three operators will now be described:

Oglebay-Norton & Co.

Oglebay-Norton & Co. operates three underground iron mines on the Menominee range. One of these mines was closed in the fall of 1930, but all three were in operation during 1929. These mines are the Berkshire and the Brule mines of the Brule Mining Co., and the Bristol mine of the Bristol Mining Co. Each of the subsidiary companies has a safety engineer. As methods are similar at each mine, description of the safety organization will be confined to that of the Brule Mining Co.

Berkshire Mine

The Berkshire mine produced 224,686 tons of medium hard hematite in 1929 and employed an average of 141 men - 94 underground and 47 on surface, in shops and office. Mining is by the top-slicing and caving systems, which are in general use on the Menominee range wherever the ore is too soft to mine by open stopes. A main ventilating fan is installed on the surface; part of the mine is ventilated by natural draft. Considerable trouble is experienced with black-slate fires; as soon as the black slate of the footwall is crushed and exposed to air and moisture it catches fire by spontaneous heating.

Labor turnover is at the rate of six men per month.

Labor Conditions

A mixture of all European races is employed in the following percentages:

Nationality	Per cent	Nationality	Per cent
Swedish	23.6	German-Polish	1.4
Italian	14.3	Hollander	1.4
Finnish	13.6	Tryolian	1.4
Polish	11.4	Armenian	0.7
French	5.0	Austrian-Croatian	.7
Croatian	3.6	Belgian	.7
British	3.6	Bulgarian	.7
Swede-Norwegian	2.9	Finn-Italian	.7
Austrian	2.9	German	.7
Irish	2.2	Norwegian	.7
Canadian-French	2.2	Scotch-British	.7
British-American	1.4	Swede-Irish	.7
British-German	1.4		
German-Italian	1.4		

Safety Organization

General Safety Committee.- The general safety committee convenes monthly at different properties operated by the company in Michigan, Wisconsin, and Minnesota. It is composed of the district manager, the general superintendents, superintendents, and safety engineers of all Oglebay-Norton & Co. properties in the Lake Superior district. The meeting is attended by foremen of the mine at which the meeting is held. A record is made of the proceedings, and copies are sent to all subsidiary companies affiliated with Oglebay-Norton & Co.

Central Safety Committee.- The central safety committee is the local mine safety committee. It is composed of the general superintendent, superintendent, safety engineer, the heads of all surface operating departments, and the underground foremen. The safety engineer presides at these meetings. Records of the proceedings are forwarded to all company properties.

Daily Safety Committee Meeting.- The daily safety committee meeting plays an important part in the safety organization. It is made up of the same members as the central safety committee and convenes at 7:30 a.m. daily. No permanent records are kept of the proceedings, the object being to outline each day's work so that all operations will function most efficiently and safely.

The safety engineer is regarded as the key man in bringing about cooperation between departments, and in organizing ways and means for a careful study of all conditions which might cause accidents. He has direct charge of the safety movement and works with the central safety committee and the superintendent toward the betterment of general safety conditions.

Safety Inspection

There are four separate forms of safety inspections, two of which are primarily for the purpose of building up safety consciousness among the employees, and as such are effective.

The foreman at the end of each shift makes out a safety and cleanliness report on every working place under his jurisdiction. The contents of this report are observed by the foreman on the opposite shift when he reports for duty, and the report is countersigned by him.

The safety engineer submits to the central safety committee a monthly inspection report covering every working place on the property, both surface and underground.

New underground employees are assigned to the duties of underground safety inspectors for the first day of their employment. They make a trip through the mine with either the section foreman or the mining captain, and are required to make a written report of their findings.

Men are selected periodically, and at random, from the underground forces to act as safety inspectors. They are required to make written reports. The result is that the men have been kept alert to dangers, and each man has learned to think about the hazards in his own job. It has gradually instilled the safety idea into the entire force.

Surface inspections as to cleanliness are made by the safety engineer every three months. Surface openings, such as test pits and shafts are inspected once every month. All surface openings are covered where possible and fenced off. All fences and warning signs are inspected regularly every month and a report is forwarded to all other Oglebay-Norton & Co. properties.

No regular form is used for inspections. The inspectors' reports are discussed at the central safety committee meetings, and the shift bosses' daily safety and cleanliness reports are discussed at the daily safety committee meeting every morning.

Safety Education

Bulletin Boards.- The company subscribes to a commercial bulletin board service and to the National Safety Council. The board is made up of three panels. The center one is used for pictures of current events and world news. The side panels are used for safety posters and pictures.

Foremanship Education.- No night classes have been established, but the company carries on foremanship training through the pamphlets on this subject issued twice monthly by the commercial bulletin service to which they subscribe.

Safety Education in Public Schools.- The local schools and civic organizations also carry on safety activities.

First-Aid and Mine Rescue Training.- The Berkshire mine received a 100 per cent training certificate in 1930 after all the men had been given first-aid training by the United States Bureau of Mines. When enough new men have accumulated to make a class, they are trained by the company instructor and the mine thus kept up to 100 per cent. The new men are examined for certificates by bureau instructors. The old men are also retrained at intervals of one year.

Twenty-four men have been trained in the use of mine rescue oxygen breathing apparatus. Five sets of McCaa 2-hour apparatus and two All-Service gas masks are kept on hand, also four canisters, two stretchers, four blankets, and a complete assortment of splints. The men are re-trained at intervals of several months and new men are trained from time to time.

Safety Standards

Drifting.- Timber is used in all drifting and crosscutting in the iron formation or in iron ore where the opening is larger than 4 feet wide and 7 feet high. No timber is used where the ground is hard enough to stand without support. In all doubtful ground, timber is used. The back, sides, and breast are required to be trimmed as soon as miners go back after a blast. Where timber is used, the back is required to be blocked at once, using four or more pieces of lagging, 8 feet long, to block ahead.

Raising.- All raises in ore or in iron formation are cribbed except in open stopes. Raises in black slate or grey slate are not cribbed if the slates are hard and of a suitable nature. Single cribbed raises are 4 by 3 feet, while double-cribbed raises are 3 by 9 feet. A safety chain is stretched across all raises while they are being driven. This device is a chain net formed of heavy chains of mesh large enough to allow the dirt to go through, but small enough to catch a man if he falls. After the blast, compressed air is blown for a reasonable length of time, then the miner climbs up and fastens the chain across the opening 15 to 20 feet from the back before he trims it down. Eight-foot ladders are used. This length allows the men to carry the ladders close enough to the back so that he does not have to climb on the cribbing to get up close enough to the back to trim it. The ladders afe set off to one side and raised 4 inches from the cribbing to allow foothold. Ladders are spliced together by laying a 2-inch board 2 feet long under the ladder and parallel with it. This allows the dirt to go through the splice under the ladder rungs and does not interfere with the footing when climbing. The other side of the ladder is nailed to the side cribbing. All raises are covered, both the mill and the ladder road. All manway raises have doors over the top and gates at the bottom, opening inward. The door over the timber slide in the manway raise is made of metal strips spaced about 1 inch apart to allow for ventilation. A solid door is used over the ladderway.

Blasting.- When blasting in drifts there are always two men to light the fuse. An extra carbide lamp is furnished every gang (a gang consists of two men) of miners for use when lighting fuses. In sub-drifts, one man lights while the other is present. In raises, one man lights while his partner waits for him at the bottom of the raise. All blasting is done at noon or when the miners go off shift. This is done so that the face may be cleared of gases in time to resume work, and to avoid danger due to delayed blast.

Fire Protection.- The air lines are connected to the water lines so that water can be turned into them in case of fire in a place not provided with a water line. Fire doors are placed on each level in each drift. Fire extinguishers are in readiness in the powder house, pump house, and at the stations in each level. The men are instructed as to what to do in case of fire. Maps are placed in the dry house, shop buildings, and engine house to show the position of hydrants and to give instructions in case of fire.

Safety Rules.-The safety rules of the company are being compiled and will be put in book form. All men underground and on the surface are already acquainted with the rules, and all new rules are explained to the men by the bosses.

Supervision.- The mine has a mining captain, two shift bosses, and one trammer boss, each man looking after about 35 men. One shift boss looks after the sixth level and the other the seventh. All places are visited twice a day or oftener, at least once in the morning and once in the afternoon.

Discipline.- Safety rules are enforced by reprimand and other forms of discipline. If a man violates a rule he is given a lecture the first time, if it is a minor offense. He may be given a day or more lay-off, or discharged, depending upon the nature of the violation. The man who continues to be careless in his habits is discharged.

Rewards.- If the mine has worked one month without a lost-time accident, tobacco, candy, or other similar reward is given to each man on the first of the month. When the mine has gone one year without a lost-time accident every employee is given a pocket knife with his name on it. The company has a safety trophy which is awarded to the mine that works three months without a lost-time accident. This trophy was in the possession of the Berkshire mine for the fall of 1930. If two mines have gone three months without a lost-time accident the trophy is awarded to the mine having the most man-hours. The Berkshire mine now has the "Sentinels of Safety" trophy of the national safety competition for the second consecutive year, as it was won by them in 1928 and 1929. It is worthy of note that this trophy was also won in 1927 by the Bristol mine operated by Oglebay-Norton & Co. under the same general superintendent, F. J. Smith.

Protection

Protective Clothing and Wearing Equipment.- At present all men underground at the Berkshire mine are equipped with hard hats, gloves, and goggles which are worn around the neck when not in use. All men wear hard-toed boots.

Safety belts are always worn when landing timber and other material from the raises. They are also worn wherever a man is working over an opening, or whenever it is considered necessary. These belts are tested every six months by a drop of 6 feet with a weight of 250 pounds. New ropes are put on the belts while they are being tested and again every three months. These belts are made of two thicknesses of heavy harness leather with a D-ring. A $\frac{3}{4}$-inch hemp rope is braided into a snap. The snaps and the new ropes are tested with the belts.

Mechanical Guarding.- Gates are placed at the bottom of every manway raise, also at the top of every manway raise, in cases where no doors are used. A heavy iron gate is used on the surface shaft landing and at each underground shaft landing. Forepoling is required whenever timber must be used underground in advancing openings. All slushers are guarded by $\frac{1}{4}$ by 1 inch iron frame with vertical and horizontal rollers. Hinged covers fit over the drums.

Fire Protection

Fire extinguishers are available at the following places: two at the seventh-level powder house, one each at the fifth and sixth level pump houses, and one at the fifth and sixth level shaft stations.

Many fire extinguishers are placed both underground and on the surface. The kinds and locations of these fire extinguishers are as follows:

Distribution of fire extinguishers

Number	Capacity	Extinguisher type	Place
1	1 Qt.	Pyrene	Seventh level powder house.
1	$2\frac{1}{2}$ Gal.	Soda acid	Seventh level powder house.
1	1 Qt.	Pyrene	Fifth level pump house.
1	$2\frac{1}{2}$ Gal.	Soda acid	Fifth level station house.
1	1 Qt.	Pyrene	Sixth level pump house.
1	$2\frac{1}{2}$ Gal.	Soda acid	Sixth level station house.
1	$2\frac{1}{2}$ Gal.	Soda acid	Lander's shanty.
1	1 Qt.	Pyrene	Do.
1	$2\frac{1}{2}$ Gal.	Nonfreeze	Saw mill.
1	1 Qt.	Pyrene	Sawmill motor room.
2	1 Qt.	Pyrene	Boiler room.
2	$2\frac{1}{2}$ Gal.	Soda acid	Dry.
1	$1\frac{1}{2}$ Qt.	Pyrene	Dry.
1	1 Qt.	Pyrene	Blacksmith shop.
1	$1\frac{1}{2}$ Qt.	Pyrene	Machine shop.
1	$1\frac{1}{2}$ Gal.	Soda acid	Do.
1	1 Qt.	Pyrene	Carpenter shop.
1	1 Qt.	Pyrene	Electric shop.
2	$1\frac{1}{2}$ Qt.	Pyrene	Engine house.
1	1 Qt.	Pyrene	Oil house.
2	1 Qt.	Pyrene	Engine house basement.
1	1 Qt.	Pyrene	Warehouse office area.
2	$2\frac{1}{2}$ Gal.	Nonfreeze	Do.
1	1 Qt.	Pyrene	Sample room.
1	1 Qt.	Pyrene	Laboratory.
1	$2\frac{1}{2}$ Gal.	Soda acid	Do.
1	1 Qt.	Pyrene	Office basement.
3	$2\frac{1}{2}$ Gal.	Soda acid.	Office main floor and upstairs.
2	1 Qt.	Pyrene	Garage.
1	1 Qt.	Pyrene	Transformer house.
1	$1\frac{1}{2}$ Qt.	Pyrene	Mounted on welding machine.
1	1 Qt.	Pyrene	Ford sedan.
1	1 Qt.	Pyrene	Chevrolet truck No. 1.
1	1 Qt.	Pyrene	Chevrolet truck No. 2.

Eight houses on the mining company location have fire extinguishers. These are both pyrene and soda acid. The superintendent's, engineer's, and cashier's residences have large fire extinguishers. Three sets of fire ladders are maintained by the company at one location and two at another. These are inspected regularly. There are four fire hydrants, one at the Berkshire location and three at the Lennox location. Five hundred feet of $2\frac{1}{2}$-inch fire hose and two nozzles are kept on a cart at the mine.

Medical Service

All new employees are given a physical examination before they are hired, and all old employees are given an examination by the mine doctor once a year. Any man who quits his job, or is discharged, or any man who takes leave of absence for over 30 days must be reexamined as a new employee. This does not apply to men who are laid off due to a temporary shutdown.

A new 32-bed hospital was built by the mining companies in the district in 1930 in the city of Stambaugh, Mich.

All cuts, scratches, bruises, and small injuries are treated by first-aid men or by the safety engineer in the dry house. If the injury is of such a nature that it requires the doctor's attention, the patient is sent at once to his office for treatment. It is left to the doctor to decide when a man shall return to work after an injury. An accident check-up system has been perfected whereby it is known at the end of each shift whether an accident of any kind has occurred. This system originated as a result of serious infections developing from small cuts and bruises of which the foreman had no knowledge. With this system in force, no injured employee can leave the property without first receiving medical treatment. A slip is put up in the "dry" each day with each employee's check number printed upon it. If a man receives a cut, bruise, or any small injury, he puts O after his number as he leaves at night; if he received no injuries whatsoever, he places X after his number. Whether he is injured or not he must mark the card.

Accident Reports

The doctor makes a report of the injury as to the nature and the cause. The safety department gets a report of the injury from the underground foreman, and the foreman's report is also sent to the office.

In classifying accidents, the man is paid full time for the day injured so that the day of injury is not counted as lost time.

In returning a man to work, he is often given an easy job until he is able to take over his regular task; as long as he is actually employed by the company, whether at the same job, at which he was injured or at some other job, it is not classified as lost time. The severity of the accident is determined by the doctor and it devolves upon him to determine whether the man is fit for shop work, underground work, or whether he is able to go back to work at all for some time. The injured man must present a clearance from the doctor giving him permission to return before he is allowed to work by the foreman. Severity and frequency charts are made up each month.

Hiring New Men

A new employee is given a safety talk by the safety engineer before he goes to work, and the first day the new man acts as a safety inspector, as mentioned above. While he goes through the mine with the foreman, he is given instruction on safety methods, safety rules, and other work. The new man then makes out a report on the condition of the mine, relative to safety and cleanliness, as well as making any suggestions or advancing any ideas which he may have along these lines.

Checking up Mental and Physical Fitness for Job

No new men, or men not known well enough by the foreman, are given hazardous jobs. Where a job is extra hazardous, a boss is always present until the job is completed.

The boss checks up on a man's mental and physical fitness for extra hazardous jobs by his personal observation of him, through contact with him day after day, and by his habits, especially as to whether he is careful or careless. He also judges by the condition of his working place, by his actions, thoughts, and ideas, and by his experience in the line of work for which he is being considered.

Compensation and Welfare

Compensation is covered by the provisions of the State Compensation Law of Michigan, which allows compensation after absence of seven days due to injury, the amount varying with the nature of the injury. The company has no special pension fund or death fund. It is self-insuring under the compensation law and sets up a reserve to compensate for accidents by charging itself a certain per cent of the payroll each month.

Inasmuch as the company is self-insuring, any reduction in the accident rate has a direct effect upon the insurance rate. If the percentage of the total payroll is not large enough to set up a reserve, it is increased.

Table 4 following is the safety record of the Berkshire mine during the last 11 years.

Table 4.- Safety record of the Berkshire mine, 1920-1930

Year	Average men	Total hours worked	Total shifts worked	Fatal accidents	Nonfatal lost-time accidents: Compensable	Nonfatal lost-time accidents: Noncompensable	Slight injuries no lost time	Days lost in nonfatal accidents: Compensable	Days lost in nonfatal accidents: Noncompensable	Total lost-time and fatal accidents per 100 men employed	Frequency rate, lost-time and fatal accidents per million man-hours
1920	185	475,336	57,228	0	18	30	31	1827	88	26.0	101
1921	107	190,631	22,951	0	11	10	4	152	32	19.6	110
1922	124	312,937	37,676	0	18	17	14	340	47	28.2	112
1923	166	427,527	51,472	0	17	20	23	682	51	22.3	86.5
1924	163	415,051	49,970	0	16	22	19	453	69	23.3	91.5
1925	180	458,275	55,174	1	17	22	36	723	60	22.2	87.3
1926	184	467,974	56,259	0	8	13	67	362	27	11.4	45.0
1927	186	471,916	56,897	0	3	0	55	113	0	1.6	6.4
1928	177	403,487	48,632	0	0	0	40	0	0	0.0	0.0
1929	141	345,695	41,612	0	0	0	44	0	0	0.0	0.0
1930	132	283,807	34,110	0	0	0	32	0	0	0.0	0.0

Table 5 lists minor no-lost-time accidents due to various causes during the years 1928, 1929, and 1930:

Table 5.- Minor no-lost-time accidents in the Berkshire mine during a three-year period

	1928	1929	1930
Fall of rock or ore	8	5	7
Rock or ore at working place or chute	7	5	3
Timber or hand tools	7	11	10
Haulage	3	-	-
Machinery	2	2	-
Overcome by gas	1	-	-
Nails and splinters	2	2	2
Electricity	1	-	-
Other causes	2	5	6
Fall of person	4	5	1
Dirt in eye	3	9	3

M. A. Hanna Co.

The M. A. Hanna Co. operated six underground mines on the Menominee range in 1929 and 1930 and produced a large part of the total output of iron ore for the range. Three of the Menominee range mines produce hard ore; the others produce soft ore. In the whole Lake Superior district this company operated 16 mines in 1929, eight being underground and eight open pit. An average of 720 men were employed in the open pits and 821 in the underground mines. Two of the underground mines are equipped with fans for main mechanical ventilation; the other six depend upon natural draft.

Labor Conditions

Of a total of 1,541 men required for the 16 mines, it was necessary to hire 148 new men per month, an average of nine at each mine. When the seasonal nature of open-pit work is considered, this is a comparatively low labor turnover.

The nationalities employed by this company in the Lake Superior district were as follows:

Nationality	Per cent	Nationality	Per cent
Finnish	18.1	French	3.5
Italian	13.7	Norwegian	3.1
Swedish	13.6	Irish	2.1
Austrian	11.0	Serbian	2.1
Polish	8.5	Scotch	1.0
British	5.9	Belgian	0.9
Croatian	4.7	Bulgarian	0.9
German	4.0	Greek	0.6
Slovanian	3.9	Miscellaneous	2.4

Of these various nationalities, 45 per cent are American born.

Safety Organization

The plan of the safety organization is not elaborate. A full-time safety engineer is employed whose duties are to make safety inspections, conduct safety meetings, and train mine rescue crews. This plan has been in effect for 15 years.

Safety Committees and Safety Meetings.- There are no standing safety committees. However, a general meeting of the superintendents, mining captains, shift bosses, mechanics, electricians, and other foremen is held at the Michigan and Minnesota district offices once a month. A meeting of all employees is also held once a month at each of the various properties.

The lost-time accidents are read and discussed at these meetings. In addition, there is a discussion of accidents which have not involved lost time, discussion of safety rules, safety suggestions, safety clothing, unsafe practices, accidents at mines of other companies, new safety kinks and devices. Outside speakers are also brought in to talk before these meetings, and salesmen from the various safety equipment companies are invited to display and talk about their goods. There have been talks by oxyacetylene welding experts and moving picture displays of safety and other educational files. Safety suggestions are requested from those attending both the general meetings and the workmen's meetings. Various safety committees make their report at the general workmen's meetings.

The number of safety meetings of employees held during 1929 at the various mines as as follows:

Mine	Meetings	Mine	Meetings
Homer	12	Mesabi Chief	4
Harold	11	Susquehanna	4
Zimmerman	11	La Rue	2
Rogers	10	Richmond	1
Wabigon	9	South Uno	1
Hiawatha	7	Maroco	1
Loretto	7	Florence	0
Bates	5	Total	85

Safety Education

Bulletin Boards.- The bulletin boards of the Elliott Service are used at all properties. In addition to the Elliott Service there are also displayed National Safety Council bulletins, local posters, honor rolls, announcements, and newspaper clippings of accidents at other companies' mines. When a man breaks his goggles with the saving of his eyes the goggles are displayed as an inducement to all men to wear their goggles.

Mine Rescue Training.- The company has eight McCaa apparatus and six All-Service gas masks available for use in case of mine fires. Forty men have been trained in the use of mine rescue breathing apparatus at the five properties in the Iron River district of Michigan, or eight men at each property. These men are re-trained every three months by the company safety engineer.

Community Meetings.- Some safety education is done in the public schools through the cooperation of the company. Other educational methods are the community safety meetings. A safety talk is given at these meetings which is followed by safety movies and other educational films and a two-reel comedy. As a rule a 5-piece orchestra is furnished to give music, and after the meeting there is dancing and a lunch. Twenty community meetings were held in 1929 at the various properties as follows:

Mine	Meetings	Attendance
Mesabi Chief	4	1250
Harold	3	850
La Rue	2	500
Loretto	2	400
Richmond	2	400
Rogers, Hiawatha, and Bates	1	400
Wabigon	2	375
Zimmerman	2	300
Wakefield	1	175
Susquehanna	1	175
Total	20	4825

Safety Inspection

The safety engineer inspects all properties regularly and continuously. Inspection is made of the surface and underground, during which particular attention to the condition of guards for machinery, loose material, safe and unsafe practices, observance of safety regulations by the men, the regular inspection of the hoisting cable, the monthly testing of safety catches on the cages, inspections for fire hazards, and to seeing that second exits are in good condition and that necessary precautions are taken to protect the workmen.

A safety committee of workmen inspects each property each month, making a report at the general safety meeting. No special report forms are used. A weekly report is sent to the general manager, assistant general manager, general superintendent, claims manager, and Cleveland office of the company.

Safety Standards for Operation and Safety Suggestions

As a result of safety meetings, a number of safety suggestions have been made and collected by the company. The safety engineer does not collect the minutes of all safety meetings or the reports of all mine safety committees,

but even with incomplete records he has a total of 677 recommendations made by employees at the various properties. These serve to show the importance of having mine safety committees and holding regular monthly safety meetings of all employees. These safety suggestions are under the following classifications:

Safety suggestions.
Suggestions for sinking safely.
Suggestions for drifting safely.
Safety suggestions for miner in driving raises.
Safety suggestions for tramming.
Safety suggestions for stope mining.
Accident prevention in slices.
Safety suggestions for men working around shovels.

Safety Rules

The safety regulations of the company have been issued in three sections. The first is for the surface - mechanical and electrical; the second for underground mining; and the third for open-pit mining. These books are given to the workmen when they are employed.

Supervision

The average number of men assigned to a shift boss is 34, a number small enough to allow him to visit each man at least two times per shift.

Rewards

Besides the community safety rallies, followed by the lunch, there are occasional safety dinners are given for the foremen. Occasional prizes are given, such as pocket knives, turkeys, and match holders.

Protection

Hard hats are required at all underground properties. This rule is even extended to visitors, none of them being allowed to go below the collar of the shaft without wearing a hard hat. Goggles are furnished to all men who need them. Hard-toed boots are not required, quite likely due to the difficulty of getting men fitted to the hard-toed boots which have been available. However, the men are encouraged to wear them.

Miners on the benches along the sides of open stopes are required to wear a safety belt and rope at all times. All openings are protected by gates. The gates at the shaft are 6 feet high at all levels. Temporary forepoling is employed in driving drifts. Slushers or scraper hoists are completely guarded both from flying particles from the rope and from getting a hand caught between the rope and the rod. Both pyrene and soda-acid fire extinguishers are used.

Medical Service

A system of physical examinations for both new and old employees has been established by the company. Hospital and dispensary service is available to the men. For slight injuries, first aid is given at the mine, and if the injury is considered severe enough the man is then sent to the doctor.

Accident Reports

The first report of an accident is made by the foreman. The doctor also makes a report of the injury when the man reports to him for treatment. The men are paid full time for the day on which they are injured, so the day of injury is not classed as lost time. If a man fails to return to work the day after the accident it must be classed as a lost-time injury.

Hiring New Men

When a man is hired he is given his first safety instruction by the foreman who hires him. The foreman explains the company's safety policy and cautions him. He is then given a rule book and told to comply with all safety rules.

Compensation and Welfare

The company is self-insuring. In addition to the compensation which the law requires, all employees carry group insurance through the mining company, the minimum being $1000, which is paid in full at death.

Pickands, Mather & Co.

Pickands, Mather & Co. produces iron ore at eight underground mines on the Menominee range of Michigan. During 1929, six of these mines were operated with a total production of 1,381,531 tons and an average of 980 employees. Three mines operated during 1929 have a main mechanical ventilation system, while the other three are ventilated by natural draft.

Labor Conditions

The percentage of nationalities employed is as follows:

Nationality	Per cent	Nationality	Per cent
American	37	German	1
Italian	16	Croatian	1
Finnish	12	Bulgarian	.7
Polish	9	Belgian	.5
British	5	Lithuanian	.5
Austrian	5	Dutch	.3
Swedish	4	Manx	.3
Canadian	2	Russian	.3

The labor turnover was at the rate of 2 per cent per month, or 20 new men were hired monthly.

Safety Organization

The general safety organization of Pickands, Mather & Co. is directed from the Cleveland office by M. D. McIntyre and Charles F. Haselton. The former, the head of the safety department at Cleveland, divides his time between safety work and handling safety insurance of all kinds. There is a range safety inspector for each of the three Lake Superior districts comprising the Minnesota ranges, with headquarters at Hibbing, the Gogebic and Marquette ranges, with headquarters at Ironwood, Mich., and the Menominee range, with headquarters at Caspian, Mich. On all ranges there are district safety inspectors working under the range safety inspectors.

Safety Committees and Safety Meetings.- Safety meetings are held quarterly. At these quarterly meetings all foremen, shift bosses, mining captains, mining engineers, superintendents, and general superintendents are present. The meeting is presided over by the range safety inspector. Accident records and statistics showing the causes of accidents, compensation costs, disability records, and the accidents themselves are discussed. Sometimes a group dinner is served at these meetings. At other times a smoker is held. Workmen's meetings have been held in the past, but they have been discontinued in recent years.

Safety Education

Bulletin boards are placed in various plants and are posted with daily bulletins. In addition to these bulletins there are also homemade bulletins on safety, sports, monthly accidents, reports, and other topics.

First-aid classes are held once a month at each mine.

Mine rescue training is given to about 40 men once a month. Twenty-five sets of apparatus are available.

Foremanship classes are held once a month.

The company cooperates with the schools in their safety educational work by furnishing safety bulletins.

Inspection

Safety inspections are made by the range safety inspector, the underground safety inspector, mining captains at each mine, the superintendents, assistant general superintendent, the general superintendent, and every official. These inspections are made every day. A report form is used for the hoist inspection and the monthly summary of safety inspections.

A recommendation slip is made out for each item to be corrected, following each inspection. These slips go to the responsible foreman and to the superintendent and are closely followed up. They are numbered consecutively and are carried as "uncompleted" until the condition is actually corrected.

Rules

Safety rules are furnished in the form of a book of rules for the foremen. There were formerly safety rules for workmen, but these have been discontinued. The company does not require any examinations on safety rules.

Discipline

Safety standards are enforced by a warning for the first offense, a lay-off for the second offense, and if there is a third infraction of the rules, the offender is dismissed.

Rewards

Competition for an annual prize is held within six groups of mines operated by this company in the Lake Superior district. Each employee of the winning mine in each group receives a prize. Any mine with a no lost-time record for the year is considered as tied for first place and is awarded the prizes. On the Menominee range there are two competitive groups, and in 1930 the Caspian mine and the West Vulcan mine were the winners. The Gogebic-Marquette district has three groups and Minnesota has two.

In addition to the yearly competitions, individual awards are made for monthly no disability records at each mine. These awards are based upon a cumulative credit system. When a mine goes through a month without disability, each employee receives a cash credit of 10 cents. If this is followed by a second month of no disability, 20 cents more is credited to each man. In the underground operations, for the third and each subsequent consecutive no-disability month 30 cents is credited. In the Minnesota open-pit operations, due to the better accident experience, the rate remains at 20 cents for the second and subsequent consecutive months.

These sums accumulate to each man's credit as long as there are no-disability months. The total credits at the end of the month are as follows:

Credits for no-disability months

Months	Underground	Minnesota open pits
1	$0.10	.10
2	.30	.30
3	.60	.50
4	.90	.70
5	1.20	.90
6	1.50	1.10
7	1.80	1.30
8	2.10	1.50
9	2.40	1.70
10	2.70	1.90
11	3.00	2.10
12	3.30	2.20

When a lost-time accident occurs, a prize for the no-disability period is decided upon and each employee in the mine receives his award. After this the accumulation of credits starts again.

The prizes are based upon the amount of credits accumulated and are given in the form of merchandise of various kinds, such as blankets, pocket knives, wool socks, fishing tackle, and other useful things. Inasmuch as the company can purchase wholesale and gives the men the benefit of the wholesale price, the amount of the credit represents more in actual goods than the money would in the man's pocket if he were to purchase his own prize at retail cost.

Protection

Wearing Apparel.- Hard hats, hard-toed boots, goggles, and gloves are worn.

This company has so many cases on record in which hard hats saved head injuries on the Menominee range that it would make a long list. The safety engineer stated that there were nine cases within five months which he learned of just by casual inquiry, and that there must be many more of which he had not heard. There was one case where a 5-pound plumb-bob dropped almost 20 feet in the shaft and punctured the hard hat, but only slightly injured the man's head. There was another case where a man working in a shaft was knocked flat by falling material which hit his head, but aside from having a stiff neck he was not hurt. At first there were many cases where the men did not want to wear hard hats, but the company insisted and they were later saved from a serious accident. No opposition is now evident.

This company had three cases where hard-toed boots saved toe injuries. In one case the boot was dented by timber falling from a truck, but the man's toe was saved.

Three cases are reported where men saved injury to their eyes by the use of goggles. These men were breaking chunks on stock piles with a sledge hammer. Their eyes were protected with glass goggles. The glass in the goggles was shattered, but no serious injury resulted to the eyes. In one case a pair of goggles with aluminum cups was badly smashed. The glass was shattered and some pieces entered the workman's eyes. The aluminum cup holding the glass was completely crushed. The injured man's face had the imprint of the goggle on it and the eye was badly bloodshot. There is no doubt that the man would have lost his eye completely if he had not been wearing the goggles. The company has introduced screen goggles in all its underground mines but up to this time has not had much experience with them.

Guarding

Safety rope and belts are used wherever a man is likely to be injured by a fall. Gates are used at raises. Temporary forepoling is used in drifting, and standard slusher guards are employed on all mechanical loading slushers. Pyrene fire extinguishers are placed near all electrical motors. Other fire extinguishers used are Phister soda, soda-acid, and Phomene.

Medical Service

New employees are given physical examinations and each employee is re-examined every year. Hospital and dispensary service is available to the men. All slight injuries are given first-aid treatment and the injured man is then sent to the doctor.

Accident Records

Reports of the injury are made by the doctor, by the foreman, and by the safety department. A man is paid full time for the day he is injured, the day of injury not being counted as lost time.

If a man is willing to take a less arduous job before he is able to return to his regular job and can earn his daily wage at such occupation, he is returned to work. This is entirely up to the workman, with the permission of the doctor, however, and works to the man's advantage. The injury is not classed as lost time as the man is returned to work, no matter if he has not been returned to his regular job. This policy has more advantages than disadvantages, although it is the subject of much controversy. By keeping a man mentally occupied, his mind is kept off his injury and his recovery is hastened. There is no opportunity to develop a neurosis, or a hallucination that he is still sick. He makes more money than he would by drawing compensation alone.

As permission to return to work is based on the doctor's decision and the man is under the doctor's observation till fully recovered, essentially all valid objections to this policy are removed and the good ones remain. The confidence of the man in his company, that they will back him up when he is injured, is worth much to the morale of the force. This practice has a further actual tendency to reduce the number of accidents, because it serves to show the men how thoroughly the management backs up its safety work and protect the men from injury.

The doctor makes a report on the severity of the accident in the form of an estimate of the time the man will be away from work. If there is any doubt, as to the time required for recovery the workman is given the benefit of it.

Placing Men on Job

The company tries, as far as possible, to give men work which they are mentally and physically capable of doing, avoiding the placing of men with serious defects in hazardous positions.

Table 6 gives the combined safety record of all Pickands, Mather & Co. mines on the Menominee range, and presents details concerning each mine.

Table 6.- Combined safety record of Pickands, Mather & Co. mines on the Menominee range

Year	Average men	Total hours worked	Accidents			Frequency rate, fatal and lost-time accidents per million man-hours worked
			Fatal	Lost-time	No disability	
1920	659	1,667,848	1	76	246	46.2
1921	304	771,559	1	32	63	42.8
1922	635	1,617,957	0	95	281	58.7
1923	1141	2,737,551	1	134	302	49.3
1924	1037	2,692,966	3	157	333	59.4
1925	1229	3,177,892	3	209	509	66.7
1926	1134	2,917,703	6	144	450	51.4
1927	1114	2,829,361	0	149	302	52.7
1928	940	2,313,452	4	79	215	35.9
1929	1009	2,453,370	4	71	294	30.6
1930	944	1,982,663	1	17	200	9.0

Per cent of No-Disability Months to Mine Months Worked

Year	Mine months worked	No-disability months	Percentage of possible perfect record
1927	108	41	38.0
1928	96	53	55.2
1929	96	53	55.2
1930	96	76	79.2

Individual mine safety records of Pickands, Mather & Co. mines on the Menominee range

BENGAL MINE

Year	Average men	Total hours worked	Accidents			Frequency rate, fatal and lost-time accidents per million man-hours worked
			Fatal	Lost-time	No-disability	
1920	208	525,850	0	31	71	59.0
1921	30	77,117	0	6	3	77.8
1922	116	296,089	0	24	56	81.1
1923	178	455,087	1	20	60	46.1
1924	132	337,255	0	25	66	74.1
1925	146	372,770	0	25	90	67.1
1926	99	251,682	0	10	55	39.7
1927	125	314,969	0	33	31	104.8
1928	100	251,081	1	19	28	79.7
1929	107	253,822	0	14	38	55.2
1930	99	78,209	0	2	4	25.6

Individual mine safety records of Pickands, Mather & Co. mines on the Menominee range - Continued

JUDSON MINE

Year	Average men	Total hours worked	Accidents: Fatal	Accidents: Lost-time	Accidents: No-disability	Frequency rate, fatal and lost-time accidents per million man-hours worked
1920	6	14,821	0	1	3	67.5
1921	65	163,926	1	7	16	48.8
1922	119	304,139	0	10	29	32.9
1923	159	407,301	0	6	34	14.7
1924	205	523,490	0	20	46	38.2
1925	168	429,794	2	25	57	62.8
1926	129	325,998	2	22	34	73.6
1927	134	336,872	0	16	36	47.5
1928	113	283,550	1	10	28	38.8
1929	128	310,037	0	13	46	41.9
1930	112	235,500	1	3	40	17.0
BUCK MINE						
1920	26	66,427	0	2	4	30.1
1921	138	350,541	0	15	31	42.8
1922	204	518,623	0	32	116	61.7
1923	154	392,455	0	17	47	43.3
1924	91	232,595	0	5	28	21.5
1925	139	355,036	0	15	38	42.2
1926	127	319,961	0	24	31	75.0
1927	127	322,382	0	12	31	37.2
1928	68	173,026	0	9	16	52.0
1929	108	224,268	1	14	34	66.9
1930	98	207,872	0	3	12	14.4
WARNER MINE						
1920	118	297,526	0	12	17	40.3
1921	42	106,370	0	2	9	18.8
1922	69	175,984	0	4	35	22.7
1923	102	261,600	0	11	35	42.0
1924	78	198,429	1	11	22	60.5
1925	123	313,389	0	20	29	63.8
1926	119	301,830	3	12	33	49.7
1927	118	301,808	0	18	34	59.6
1928	95	238,781	0	1	22	4.2
1929	99	251,219	0	2	13	8.0
1930	101	239,581	0	3	13	12.5
CASPIAN MINE						
1920	301	763,224	1	30	151	40.6
1921	29	73,605	0	2	4	27.2
1922	127	323,122	0	25	45	77.4
1923	209	533,086	0	36	75	67.5
1924	168	431,453	1	19	63	46.4
1925	164	420,132	0	27	63	76.0
1926	149	377,550	0	19	55	50.3
1927	143	364,100	0	26	32	71.4
1928	132	332,953	0	21	22	63.0
1929	152	361,625	0	16	52	44.2
1930	142	307,662	0	2	37	6.5

Individual mine safety records of Pickands, Mather & Co. mines on the Menominee range - Continued

EAST VULCAN MINE

Year	Average men	Total hours worked	Accidents: Fatal	Accidents: Lost-time	Accidents: No-disability	Frequency rate, fatal and lost-time accidents per million man-hours worked
1920	-	-	-	-	-	-
1921	-	-	-	-	-	-
1922	-	-	-	-	-	-
1923	16	28,834	0	0	8	0.0
1924	108	277,967	1	27	49	100.7
1925	164	418,560	0	36	79	86.0
1926	172	448,353	1	11	91	26.8
1927	191	492,434	0	2	62	4.1
1928	206	484,030	0	0	39	0.0
1929	187	482,582	1	2	42	6.2
1930	189	437,970	0	4	46	9.1

JAMES MINE

Year	Average men	Total hours worked	Accidents: Fatal	Accidents: Lost-time	Accidents: No-disability	Frequency rate, fatal and lost-time accidents per million man-hours worked
1920	-	-	-	-	-	-
1921	-	-	-	-	-	-
1922	-	-	-	-	-	-
1923	-	-	-	-	-	-
1924	-	-	-	-	-	-
1925	108	278,317	1	38	62	140.1
1926	126	318,496	0	34	56	106.8
1927	97	246,389	0	36	37	146.1
1928	71	182,404	2	15	24	93.2
1929	74	167,882	1	5	24	35.7
1930	72	157,467	0	0	16	0.0

WEST VULCAN MINE

Year	Average men	Total hours worked	Accidents: Fatal	Accidents: Lost-time	Accidents: No-disability	Frequency rate, fatal and lost-time accidents per million man-hours worked
1920	-	-	-	-	-	-
1921	-	-	-	-	-	-
1922	-	-	-	-	-	-
1923	324	659,188	0	44	42	66.7
1924	255	691,777	0	50	59	72.3
1925	217	589,894	0	23	91	39.0
1926	213	573,833	0	12	95	20.9
1927	178	450,347	0	6	39	13.3
1928	156	367,627	0	4	36	10.9
1929	154	401,935	1	5	45	14.9
1930	132	318,402	0	0	32	0.0

CONCLUSION

Safety organization must be adapted to the needs and peculiar working conditions of the company. The essential requirements of any safety organization, however, are that it shall establish a means of organized inquiry into underlying causes of accidents, and provide authority to be a means of mitigating the causes which can be eliminated.

The mine-safety program should provide, as a standby for emergencies, first-aid treatment and medical service as well as equipment and trained men for rescue and recovery work. The safety department should have at its disposal a complete system of reports and tabulation service to enable it to determine quickly the facts of an accident. These facts may be, among others, causes, location, time of day, and frequency of accidents to individuals or to groups working under unusual conditions. The safety engineer himself usually should not make the actual tabulation of statistics because his time can be used to better advantage. A good system of reports as to accidents, including lost-time and no-lost-time and near-injury accidents with tabulation under various headings, is as much the guide of the safety department as the accounting system is to the financial and managing department.

When this preliminary organization has been set up, the real work of accident prevention begins, embracing as it does a number of phases such as inspection, protection, and guarding, setting up safety standards and regulations, formulating safety rules, enforcing rules and standards through good supervision and wholesome discipline, and raising the safety consciousness of the workmen by safety education. The end to be accomplished is that the workman may be permitted to live with all his limbs and faculties to enjoy his life with his family, and that his wife may keep her helpmate to round out her life with happiness rather than merely existing after his death from accident or infection, with the empty economic comfort furnished by compensation.

I. C. 6522

October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METHOD AND COST OF QUARRYING, CRUSHING, AND GRINDING LIMESTONE AT THE CATSKILL PLANT OF THE NORTH AMERICAN CEMENT CORPORATION, CATSKILL, N. Y.

BY

W. J. Fullerton and Albert W. Cox

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METHOD AND COST OF QUARRYING, CRUSHING AND GRINDING LIMESTONE AT THE CATSKILL PLANT OF THE NORTH AMERICAN CEMENT CORPORATION, CATSKILL, N. Y.[1]

By W. J. Fullerton[2] and Albert W. Cox[3]

INTRODUCTION

This paper is one of a series being prepared by the U. S. Bureau of Mines describing mining and milling methods and costs at cement-plant quarries throughout the United States, and deals directly with methods and costs at the quarry and mill of the North American Cement Corporation at Catskill, N. Y. The process of manufacturing cement is described only as far as the final grinding of the raw materials.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent local operating expenditures only and not total production costs. It is recognized that publication of total production costs may in many instances cause embarrassment to individual producers, as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations may ensue.

The authors wish to acknowledge the assistance of the following persons who helped in the collection of information: R. W. Jones, mining engineer of Catskill, N. Y.; G. A. Witte, assistant general manager; H. F. Kichline, chief chemist; and C. A. Hartman, quarry foreman.

HISTORY

The first prospecting on the property was done in 1914 and 1915, and quarrying operations were started in February, 1916. During the early opera-

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6522."

2 - One of the consulting engineers, U. S. Bureau of Mines, and engineer of the North American Cement Corporation.

3 - One of the consulting engineers, U. S. Bureau of Mines, and experimental engineer of the North American Cement Corporation.

tion of the quarry, drilling was done by tripod drills and all stone was loaded by hand. Transition to the modern methods of quarrying, to be described subsequently, was made gradually.

GEOLOGY

The Devonian beds in which the quarrying operations are conducted are folded into a series of anticlines and synclines, with the folds strongly developed along a general north-south strike. Another series of folds has an east-west strike, but is not so pronounced. The formations are disturbed by a series of overthrusts, rather closely spaced and having an easterly dip. The overburden varies from 0 to 5 feet in thickness and consists of glacial sand and clay deposits with a small amount of siliceous material from the Oriskany.

The limestone is of the following designations: Esopus, Alsen, Becraft, New Scotland, and Coeymans. The Becraft beds are high in calcium carbonate, and quarrying operations are conducted largely in them, with the method of quarrying determined by their position. Figure 1 shows an elevation of the quarry with the limits of the present working face.

The vertically dipping Becraft stone at the east end of the quarry is difficult to drill due to seams, faults, and an occasional small cave. Toward the west, where the dip is not so steep, drilling is less difficult, although the greater hardness of the New Scotland stone underlying the Becraft at the western end of the quarry slows up drilling. About one hole out of ten is lost, due to broken drills, seams causing the drill to lose direction, etc. Most of the trouble is encountered in the vertically folded stone at the east end of the quarry.

PROSPECTING AND SAMPLING

As stated, the first prospecting on the property was done in 1914. Holes were put down with a shot drill and the core samples obtained were analyzed by R. W. Hunt & Co. of New York.

Prospecting for the second (North) quarry was done in 1920. In this instance well drills were used and sludge samples taken at every 4 feet of depth. The holes were spaced about 30 feet apart across the rock fold and 100 feet apart along the fold, and varied in depth from 60 to 100 feet.

Prospecting for the present (South) quarry was started April 17, 1925, and is still being carried on at intervals when drilled reserves require extension. Thus far, 85 well-drill holes ranging from 60 to 110 feet deep have been sunk. The spacing of these holes is irregular and was made to suit the known approximate strike of the rock folds. As a general thing the position of holes was restricted to points where the Becraft limestone was not exposed or where there was doubt about the grade of rock underlying the surface. Samples of drill-muds are taken at 5 foot intervals and are analyzed for calcium-carbonate content, with complete analyses when considered necessary.

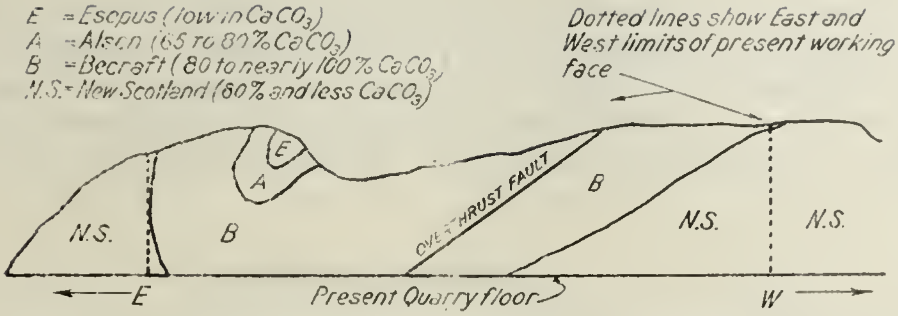

Figure 1.- Elevation of quarry, looking south

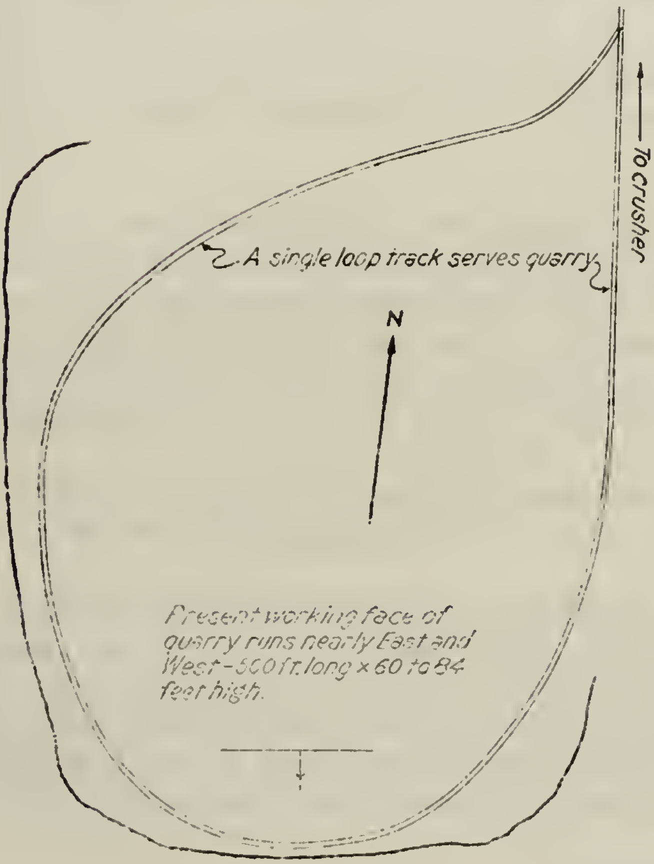

Figure 2.- Plan of quarry and track

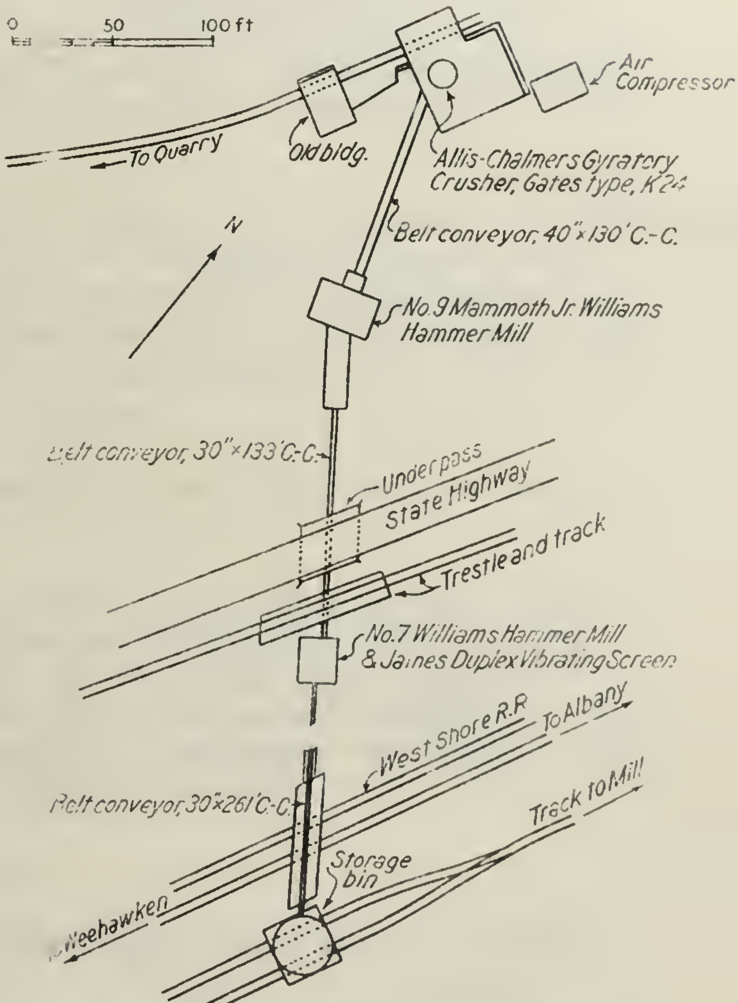

Figure 3.- Layout of crushing plant

Prospecting is done when convenient as a matter of routine and is kept about 400 feet in advance of the quarry face.

After major jobs of prospecting, estimates of reserves are made by plotting the holes to scale on cross-section paper, and considering dip, average thickness, and length of formations, and calculating the contents of the proved ground in cubic yards. The tonnage is figured by assuming a weight of 125 pounds per cubic foot. For convenience, the stone is considered to be of two grades; one containing 80 to 100 per cent of $CaCO_3$, and the other from 60 to 80 per cent of $CaCO_3$.

QUARRYING METHODS

Figure 2 shows a plan of the quarry and track. The quarry face is approximately normal to the strike of the folding. The present working face runs nearly east and west, and is about 500 feet long by 60 to 84 feet high. The quarry is served by a single-loop standard-gage track. The primary crusher is situated approximately north of the working face and about half a mile distant.

The light overburden, varying from 0 to 5 feet, is not removed by stripping, but is shot down and handled with the stone. It is highly siliceous, and silica is a necessary component for the cement mixture. The percentage composition of the overburden is as follows: CaO, 5.5; SiO_2, 59.8; Fe_2O_3, 6.4; Al_2O_3, 14.3; MgO, 2.9; loss on ignition, 10.0; undetermined, 1.1. The ratio of overburden to stone is about 1 to 50 by weight.

There is natural drainage from the quarry, so no provisions for taking care of water are required.

Quarrying operations are confined to materials suitable for the cement mixture, so there is no waste from the quarry, all the stone going to the mill. The output varies from 1,500 tons per day for full mill operation to 1,000 tons per day for restricted operation. The overburden included in this output is about 2 per cent of the total tonnage. The quarry operates in the daylight hours only and averages about 10-1/2 hours a day.

Drilling

Three Loomis "Clipper" drills, of traction type, are used for primary drilling. A description of them follows:

1 No. 4ET, type AVE, powered by a 15-hp. electric induction motor;

1 No. 4ET, type AVE, powered by an 11-hp. electric induction motor;

1 No. 4GT, type JS, powered by an 18-hp. Cook gasoline engine.

The electric drills are liked best because they are easily started in cold weather and because it is not necessary to haul gasoline and water to them. But since it is inconvenient to operate them at points far from the electric power lines, the gasoline drill is used for prospecting. Comparative figures

for operating costs of the different drills are not available, but the following figures of performance during 1930 should be interesting:

Comparison of drill performance, 1930

Drill	Total feet drilled	Hours operated	Feet per hour
15-hp. electric	5762	1592	3.62
11-hp. electric	3927	1514	2.59
18.hp. gasoline	2806	1119	2.51
Total	12495	4225	Average 2.96

The bits used with the drills are of the Gill type, and vary in diameter from 5-1/2 inches for the smallest to 6-1/4 inches for the largest; they are all about 4 feet long. The holes drilled are about 6 inches in diameter and extend to a point 6 or 8 feet below the quarry floor. The holes for primary blasting are spaced about 30 feet back from the quarry face and 16 to 18 feet apart. The spacing is controlled by the required fragmentation, the height of face, and the hardness of rock. The present practice in spacing holes was developed by experimenting over a period of about 18 months, during which time six primary blasts were set off. Distances between holes ranging from 16 to 22 feet and distances from face to holes ranging from 20 to 32 feet were tried, one set of measurements being used for one blast so that there was no chance for confusion through using several different spacings in one blast. After calculating the tonnage shot down against the fragmentation and ease of cleaning up the shots, the present spacings were adopted as standard.

A 7-inch iron pipe 6 or 7 feet long is used to case through the light overburden during drilling.

Secondary drilling for breaking bowlders is done by jackhammer drills, using 7/8-inch hexagon bits, varying in length from 2 to 14 feet. Compressed air having a pressure of 70 pounds per square inch is used.

Blasting

Twenty or more holes are loaded for each primary blast. A crew of 10 men handles the routine of loading, which is about as follows: 3 men truck dynamite from the magazine to a position near the holes, 3 men carry dynamite and earth for tamping to the holes, and 4 men load the holes. It takes about one-half hour to load each hole.

No springing of holes is done. Where the rock is hard and solid the hole may be completely filled with dynamite, but for less solid rock the dynamite is alternated with 5 or 6 feet of dirt, so placed that the dirt plugs in adjoining holes will be at different levels. At the bottom of every second hole and in all holes at the end of the straight section of the working face, where it curves from an east-west direction toward a north-south direction, 100 pounds of 75 per cent quarry gelatin is used. Clay is used for tamping in summer, and stone screenings in winter.

No. 6 Hercules exploders, with plain and covered cordeau, detonate the primary blasts. No. 6 Hercules exploders are used for secondary blasts.

The following tabulation shows the quality and quantity of dynamite used in 1930.

Total stone quarried in 1930 . . . . . 281,400 short tons

Dynamite used for primary blasting (sticks 5 x 16 or 24 in.):

	Pounds
75 per cent quarry gelatin	11,250
60 per cent quarry gelatin	30,850
60 per cent Red Cross	37,800
60 per cent Gelex A	9,050
Total	88,950

Stone blasted per pound dynamite	3.16 tons
Dynamite used for secondary blasting (sticks 1 x 8 inches) (Extra D)	9,450 pounds
Stone blasted per pound dynamite (secondary) .	29.78 tons
Dynamite used for both primary and secondary blasting	98,400 pounds
Stone blasted per pound dynamite	2.86 tons

The track along the working face and for a safe distance back along the line toward the crusher is removed before setting off primary blasts. The whole quarry force is employed in removing the tracks and replacing them later. The shovel loads the rails and the bundles of ties slung together with chain, onto the quarry cars, which are moved away to a safe distance. Following the blast and after the shovel has cleaned up the loose stone, the track is replaced.

Loading

The steam shovel is a 100-B, full-circle, caterpillar-type Bucyrus with a 3-yard dipper. It loads the stone into 12-yard Western side-dump cars which are hauled by an 18-ton Vulcan steam locomotive to the primary crusher. The shovel crew consists of one operator, one fireman, and one pitman. The one shovel handles the quarry output of 1,000 to 1,500 tons per day. The steam shovel is being replaced by a Marion electric shovel, type 4160, mounted on crawling traction trucks, equipped with 34-foot steel boom, 21½-foot dipper stick and 4-cubic yard dipper, and operated by 2,300-volt, 3-phase, 60-cycle alternating current. The steam shovel is to be transferred to another quarry of the company, where the tonnage handled is somewhat less.

A shovel of the electric type was deemed desirable for several reasons, the principal ones being cheaper operation and the ease of bringing power to

the shovel and keeping the shovel in repair. The electric shovel has a 4-yard dipper, as compared with a 3-yard dipper on the shovel it is replacing. Power will be brought to the shovel by a No. 4 four-conductor Rome super-service cable, 750 feet long.

Upon arrival at the primary crusher, the loaded cars are dumped by means of a compressed-air cylinder, which hooks and lifts one side of the car, dumping the stone into the crusher. The layout of the crushing plant (not including grinding mills) is shown in Figure 3.

Transportation

The track in the quarry is laid directly on the quarry floor and is not ballasted. The men who do the secondary drilling and blasting also maintain the track. Little work is required for routine maintenance, probably only a few man-hours in a month.

Although the plan of the quarry (fig. 2) shows the quarry track as a loop, the full loop is not used at present. The locomotive approaches the shovel with empties from the east and removes these cars to the east after they are loaded. After the removal of the four loaded cars from the shovel, the locomotive must take them to the crusher and pick up empties to serve the shovel, hence the shovel is idle for about 10 minutes while waiting for the empties.

In present practice the locomotive is always on the crusher side of the loads at the shovel, but at the run-around switch at the crusher it shunts them by and getting on the shovel side of the loads, pushes them beyond the crusher, and picks up a train of four empties. The locomotive is then between the loaded cars and the empties, and starting for the shovel with the empties on the shovel side, drops the loaded cars at the crusher where a car puller operated by compressed air places the cars for dumping.

The loop track will be placed in use within a few weeks. The locomotive will then handle trains in cars of three, approaching the shovel from opposite sides alternately, and pushing a train of loads away as it places a train of empties. In this way there will be no lost time at the shovel. Three trains of three cars each will be used, and while a train is in transit, there will be a train at the shovel and another at the crusher. This change in practice will involve no changes in track layout other than some grading on the storage track beyond the crusher, so that empties can be dropped down for the locomotive.

In moving the track serving the working face, the track is taken apart in sections about 150 feet long, and the shovel lifts the track to its new position. It makes the first move of the track from its position on the quarry-face side of the track, then moves through the opening between the sections of track to the outside of the track. The work is completed from the outside, except that an opening large enough for the shovel to pass is left, and this is closed after the shovel is again between the track and face.

Crushing and Screening

The 1,500 tons of stone (plus overburden) quarried per day passes

successively through a gyratory crusher, a large hammer mill, and a small hammer mill. About 150 tons of clay is added at this point and passes through the ball mills and the tube mills with the stone, each of the two ball mills handling about half the total tonnage and each of the four tube mills handling about one-quarter of it. There is no sizing during the process, except that a screen with 5/8-inch openings removes about one-third of the discharge of the large hammer mill as undersize which by-passes the small hammer mill. However, upon completion of the new screen equipment the stone-clay-water slurry will be screened at the ball mills.

The machines used for crushing and grinding, with the approximate size of reduction and the rate of output for each, are listed below. The output figures for slurry are based on weights of dry material, and do not include the weight of water.

Machines used for crushing and grinding, showing approximate size of reduction and rate of output for each

Machines	Size of material: Entering	Size of material: Leaving	Output, tons per machine per hour
1 No. 24 Gates-type gyratory crusher	4 feet and under	8 inches and under	[1] 104
1 No. 9 Williams hammer mill	8 inches and under	1 1/2 inches and under	[1] 104
1 No. 7 Williams hammer mill	1 1/2 inches and under	5/8 inch and under	[2] 70
2 Allis-Chalmers ball mills, 8 ft. diam. 7 ft. long (preliminators)	5/8 inch and under	94 per cent passing 10-mesh sieve	[3] 31.9
4 Allis-Chalmers tube mills, 7 ft. diam. by 16 and 20 ft. long	97 per cent passing 10-mesh sieve	90 per cent passing 200-mesh sieve	[3] 15.4

Per cent passing:	No. 7 hammer-mill discharge = ball-mill feed	Ball-mill discharge = tube-mill feed
200 mesh	--	51.0
100 "	11.1	55.8
80 "	12.5	57.4
60 "	14.2	59.4
50 "	16.4	62.2
40 "	20.1	66.4
30 "	24.2	71.2
20 "	41.2	79.4
10 "	47.0	93.8
6 "	63.5	--
4 "	--	100.0
3/8 "	88.5	--
3/4 "	100.0	--

1 - Approximate average output in 1930; figures do not represent capacity of machines, but average output.
2 - Estimated average output; about one-third the material by-passes this mill through the screen.
3 - Average output in 1930.

Note: The gyratory crusher and the two hammer mills crush stone (plus overburden); the two ball mills and four tube mills grind slurry (stone-clay-water). The output figures for these grinding mills are based on weights of stone and clay without water; the proportion of stone to clay is about 9 to 1.

The output figures for the ball mills and tube mills are based on the mill hours operated and tons ground for the year 1930. The outputs per hour for these mills are not comparable to those for the gyratory crusher and hammer mills for the reason that the grinding mills operate day and night, as needed to maintain stocks of ground material for the kilns, whereas the crusher and hammer mills operate in the day only. The slack is taken up by the concrete stone storage bin after No. 7 hammer mill, by the open storage served by a Whiting electric crane, and by the preliminator (ball mill) feed bins. There is no storage between the ball mills and the tube mills, so the tube mills must operate when the ball mills operate. The operating ratio is usually two tube mills to one ball mill, but sometimes with one ball mill running, three tube mills are run in order to maintain the fineness of hard grinding material. This will account for the slight discrepancy between the total of 61.6 tons per hour for the four tube mills and the 63.8 tons for the two ball mills.

The gyratory crusher is a Gates-type Allis-Chalmers crusher, model K-24, and is driven through a belt by a 200-hp. induction motor. The larger hammer mill is a No. 9 Williams Mammoth jr., and is driven at 870 r.p.m. by a direct-connected 500-hp. slip-ring motor.

The smaller hammer mill is a No. 7 Williams New Type and is driven at 1170 r.p.m. through a direct drive by a 200-hp. slip-ring motor. This mill is preceded by a 2-deck James Duplex vibrating screen, with screens 4 feet wide and 7 feet long. The upper deck is for heavy screening and scalping, and the lower deck for producing the finished product. The stone passing through the lower deck is about one-third of the total tonnage handled by the screen, so that the material retained on the screens and going from them to the hammer mill is about two-thirds of the total tonnage. The stone leaving the small hammer mill and that passing through the screen is carried away by a belt conveyor.

Grinding

The following table gives operating data for the grinding mills during 1930 when 312,700 tons of slurry (not including weight of water) was ground.

Operating data for grinding mills, 1930

	Ball mills	Tube mills
Approximate ball load, pounds	24,000	50,000
Diameter of balls, inches	4	7/8
Total weight of balls, worn out, pounds		319,305
Total cost of balls		$13,922
Weight of balls worn out per ton of ore ground, pounds		1.021
Cost of balls per ton of ore ground		$0.0445
Cost of replaced liners	$281.00	$4,701.00
Cost of replaced liners per ton of ore ground	$0.0009	$0.0150

No ball consumption is shown for the ball mills. There was an unknown quantity of old balls used, balls that were not carried on the stores card and which were thus not charged to the ball mills when used. Hence there is no way of computing the consumption.

The cost figures represent the cost of the balls and liners only and do not include labor. Ten end liner plates, and no side liners, were put in the ball mills. Seventeen end liner plates and two complete sets of liner plates were put in the tube mills.

The two ball mills are Allis-Chalmers preliminators, 8 feet in diameter by 7 feet long. One mill has been equipped with a rotary trommel screen, the oversize from which passes to an elevator to be returned to the mill. This trommel screen will be removed, however, as two 8-foot, type 42 "Hum-mer" screens are being installed to serve the two preliminators. As mentioned before, clay and water are added to the stone as it enters the preliminators, and the "Hum-mer" screens will handle this slurry. The slurry passing through the screens will go directly to the tube mills, while the oversize will be returned to the preliminators for further grinding. Each preliminator is driven at 20 r.p.m. through a countershaft, pinion and gear, by a 200-hp. synchronous motor.

The four tube mills are of Allis-Chalmers make, 7 feet in diameter; two are 16 feet long and two are 20 feet long. Each tube mill is driven at 21.8 r.p.m. through a countershaft, pinion and gear, by a 300-hp. induction motor.

As stated, grinding of the stone with clay is done to reduce and mix the materials preparatory to burning to cement clinker. Although no grading for sizes is done, close control of the fineness of the slurry leaving the tube mills is effected by means of sieve tests made at hourly intervals. Coarse particles are detrimental to easy and complete combination of the components during burning. The fineness of the slurry leaving the tube mills is regulated by controlling the rate of their feed. The fineness of the slurry is kept at or above 90 per cent passing a 200-mesh sieve.

The chemical composition of the raw material is kept under strict control. Samples of the slurry leaving the tube mills are analyzed hourly for their calcium-carbonate content. Chemical control of materials extends from the quarry through the various plant departments, but it is not necessary to describe it here. The proper composition of the raw materials is attained by the following controls:

- Quarrying stone that is suited for the cement mixture.
- Blending of crushed stone in the open storage.
- Use of proper amounts of clay, varying the quantity to suit composition requirements.
- Blending of slurry in tanks by mechanical and air agitation.

The material-handling equipment and practice are summarized as follows:

From	Means of transportation	To
Quarry face	One 18-ton Vulcan steam locomotive, 4-wheel, saddle-tank, hauls quarried stone about 1/2 mile in 12-yard, Western, side-dump cars in trains of four cars over standard-gage track; rails, 60 pounds per yard; maximum grade, about 1 per cent	Gyratory crusher.
Gyratory crusher	One belt conveyor, with belt 40 inches wide and 130 feet between centers, conveys crushed stone	No. 9 hammer mill.
No. 9 hammer mill	One belt conveyor, with belt 30 inches wide and 133 feet between centers, conveys crushed stone	No. 7 hammer mill.
No. 7 hammer mill	One belt conveyor, with belt 30 inches wide and 261 feet between centers, conveys crushed stone	Concrete stone storage bin.
Concrete stone storage bin	One 40-ton Porter steam locomotive, 4-wheel, saddle-tank, hauls crushed stone about 4,000 feet in 40-ton steel hopper cars in trains of two cars over standard-gage track; rails, 67 and 80 pounds per yard; maximum grade, about 2 per cent	Open storage.
Open storage	One Whiting electric crane, 3 yard capacity, span 80 feet, powered by four motors, handles stone in storage and reclaims it from storage (this crane also serves cement-clinker and coal storages). It delivers to one 24-inch by 153-foot belt conveyor, which in turn delivers to the 20-inch by 220-foot belt conveyor that delivers to the ball-mill feed bins	Ball mills (preliminators).

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From	Means of transportation	To
Clay quarry	One 40-ton Porter steam locomotive (used also for hauling stone) hauls clay about 1,500 feet in 6-yard, Western, side-dump cars in trains of three cars over standard-gage track; rails 67 and 80 pounds per yard; maximum grade about 2 per cent.	American clay machine.

Note: Water is added to clay in the clay machine and the clay in suspension is pumped to storage tanks by a centrifugal pump. It runs by gravity, as needed, to the two ball mills, where it meets the stone and where more water is added. The slurry consists of about 57 per cent of stone, 5 per cent of clay, and 38 per cent of water.

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From	Means of transportation	To
Ball mills (preliminators)	Two Link-Belt chain-and-bucket elevators, with 8 by 8 by 16 inch buckets, receive the slurry (stone-clay-water) from the trough leading from the two ball mills, and deliver it to a 40-foot Smidth horizontal mechanical agitator which feeds it to the four tube mills	Tube mills.

The electric power used by the motors is 3-phase, 60-cycle, 440-volt alternating current. The air compressor serving the quarry is a 750 cubic foot Chicago Pneumatic machine driven by a 130-hp. synchronous motor.

QUARRYING AND HANDLING CLAY

Clay-quarrying operations are simple, and a few words will suffice to describe them. Clay is used as a source of silica, iron, and alumina for the cement mix. The clay is soft and plastic with no sandy material and but little loam. One 3/4-yard full-revolving, caterpillar-type, Osgood steam shovel is used. The dipper noses into the clay bank and tears out the material which it loads into 6-yard, Western, side-dump cars. The clay is hauled by steam locomotive to the 10-foot American clay machine, where water is added and the clay broken. The suspended clay is pumped to storage tanks, whence it runs by gravity to the preliminators, as needed.

SAFETY AND FIRST-AID WORK

In discussing the subject of safety and first-aid work it is difficult to separate safety activities in tho quarry from those of the plant as a whole, but the following statements are applicable to the quarry in that it is one department of the cement plant.

There is a plant safety committee, made up of all plant department heads, which meets monthly to discuss safety and first-aid matters. There is another group known as Distinguished Safety Workers Club, made up of 12 workmen who have shown outstanding interest in safety work. This group meets monthly and is presided over by the chairman of the plant safety committee. Each second Saturday a departmental safety meeting is held, at which a workman gives a talk on safety and dangerous and unsafe practices are discussed and suggestions made for their elimination. A good-housekeeping campaign, with regular inspections, is carried on in all plant departments.

Some time ago a first-aid instructor from the U. S. Bureau of Mines visited the plant and trained selected men in first aid, and the bureau first-aid methods are used at the quarry and throughout the plant. Tho men first trained in this work have in turn acted as instructors for other men of the plant organization, until now 40 of the 150 men employed at the plant have had first-aid training. There are two first-aid teams in the plant, made up of the men most proficient in this work, and it is planned to have these teams compete.

QUARRY ORGANIZATION AND SCALE OF WAGES

A list of men employed at the quarry will suffice to indicate the quarry organization and includes the following:

1 quarry foreman
3 well drillers
1 shovel operator
1 shovel fireman
1 shovel pitman
1 blaster
1 locomotive engineer
1 gyratory-crusher operator
1 jackhammer driller
1 hammer-mill operator (for 2 mills)
1 conveyor-belt operator
1 blacksmith
1 blacksmith's helper
2 car repairmen
1 night watchman

The total quarry force, including employoes of the crushing department but not those of the grinding department or clay quarry, is 18 men. The foreman is paid a monthly salary and the workmen are paid on an hour basis, their rates of pay varying from 40 cents to 73 cents, with an average of 53 cents an hour. There is no contract or bonus system used. The quarry normally is operated 10-1/2 hours a day, in daylight only, and about 270 days a year. The quarry is usually closed on Sunday and there is a shutdown of some weeks during the winter.

COSTS

The costs of quarrying stone, crushing stone, quarrying clay, and grinding stone plus clay given in the following table are based on operations for the full year 1930.

Quarry costs for dry stone. Year of 1930; stone quarried, 281,400 tons

	Per ton
All labor	$0.0988
Supplies other than power, fuel, and explosives	.0394
Electric power	.0061
Coal (shovel and locomotive)	.0166
Explosives	.0499
Stripping	- 0
Total cost	$0.2108

Cost of crushing dry stone (before addition of clay) in gyratory crusher and two hammer mills. Year of 1930; stone crushed, 281,400 tons

	Per ton
All labor	$0.0371
Supplies other than power and fuel	.0436
Electric power	.0178
Coal for locomotive hauling	.0047
Total cost	$0.1032

Cost of quarrying clay. Year of 1930; clay quarried, 31,300 tons.

	Per ton
All labor	$0.0835
Supplies other than power and coal	.0118
Coal for shovel and locomotive	.0178
Total cost	$0.1131

Cost of grinding slurry (stone plus clay). Year of 1930; material ground, 312,700 tons.

	Per ton
All labor	$0.0613
Supplies other than power	.0797
Electric power	.1398
Total cost	$0.2808

Labor for grinding includes wages of clay-machine operator, belt-conveyor operator, etc.

Man-hours per unit of production. Year 1930; stone, 281,400 tons; clay, 31,300 tons.

Quarry: Man-hours per ton of stone quarried.	0.169
Crushing: Man-hours per ton of stone crushed	.073
Clay: Man-hours per ton of clay quarried ...	.168

I. C. 6523 September, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

PYRITES

GENERAL INFORMATION

BY

Robert H. Ridgway

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I.C.6523.
September, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PYRITES[1]

By Robert H. Ridgway[2]

CONTENTS

INTRODUCTION

This circular outlines salient facts regarding the pyrites industry of the United States and the world. It is founded chiefly upon published information available in the literature of the subject. The United States Bureau of Mines has in its files additional and more detailed data upon many of the subjects presented very briefly here, and will endeavor to assist the mineral industries and the public by giving further information in response to individual inquiries. The bureau will welcome comments, criticisms, and contribu-

1 The Bureau of Mines will welcome reprinting of this paper, providing the following footnote acknowledgment is used "Reprinted from U. S. Bureau of Mines Information Circular 6523."

2 Mining engineer, common metals division, U. S. Bureau of Mines.

tions of data that may assist in making later editions of this circular of information more accurate, or may aid in developing more fully the files pertaining to pyrites.

PROPERTIES

The term pyrites is the indefinite general trade name for any of the iron-sulphide minerals containing from 25 per cent to over 50 per cent of sulphur. The minerals composing this group are chiefly pyrite, marcasite, and pyrrhotite. The name pyrite is derived from a Greek word meaning fire and alludes to the brilliant sparks produced when the mineral is struck with iron.

Pyrite (FeS_2) is one of two forms of iron disulphide. It crystallizes in the isometric system as cubes, the faces of which are often striated, octahedrons, and pyritohedrons (pentagonal dodecahedrons). The crystals are of common occurrence and are often beautifully and sharply developed. The mineral is opaque with a pale brass-yellow color and a bright metallic luster; its streak is greenish or brownish black. It has no distinct cleavage, is brittle, and breaks with a conchoidal to uneven fracture. Pyrite ranges from 6 to 6.5 in hardness, which is unusually hard for a sulphide; it scratches both glass and steel. The specific gravity varies from 4.95 to 5.10. Pyrite when pure contains 53.4 per cent of sulphur and 46.6 per cent of iron, but often it contains small amounts of arsenic, nickel, cobalt, copper, gold, zinc, tin, etc. It is insoluble in hydrochloric acid but when powdered is completely soluble in nitric acid.

Owing to its yellow color, pyrite has been mistaken frequently for gold and hence has earned the name of "fools gold." It is distinguished from gold by its hardness and brittleness, gold being soft and malleable. It is distinguished from chalcopyrite by being harder and by its pale yellow color.

Marcasite (white iron or cockscomb pyrite) has the same chemical composition as pyrite and is difficult to distinguish from the latter, especially when in massive form. It forms tabular crystals in the orthorhombic system but is seldom found in simple crystals. Owing to its multiple twinning, it usually forms groups with jagged outlines and reentrant angles which have suggested the names "cockscomb pyrites," "spear pyrites," etc. It frequently occurs in radiating fibrous masses. Its color is pale yellow to almost white and deepens on exposure. It has a metallic luster, is opaque, and gives a grayish black streak. Marcasite has the same hardness as pyrite (6 to 6.5), but the specific gravity is slightly less, ranging from 4.85 to 4.90. The cleavage is distinct and parallel to crystal faces. The mineral is brittle and its fracture is uneven. It may be distinguished from pyrite by its color, crystal form, and by the sulphur residue formed when the mineral is dissolved in nitric acid.

Pyrrhotite is a mineral of varying composition. Different formulas ranging from Fe_5S_6 to $Fe_{16}S_{17}$ have been given; they all conform to the general formula Fe_nS_{n+1}, with $Fe_{11}S_{12}$ the usually accepted formula. Pyrrhotite is regarded as a solid solution of sulphur in FeS. Pure $Fe_{11}S_{12}$ would contain 38.4 per cent of sulphur and 61.6 per cent of iron. It often contains cobalt, nickel, or copper, occasionally arsenic, and sometimes silver, gold, platinum, and some of the rarer metals. Pyrrhotite crystallizes in the hexagonal system, usually with tabular and sometimes with pyramidal crystals, but it generally occurs in massive form with granular structure. It has a brownish bronze color and metallic luster but tarnishes easily. It is softer than pyrite or marcasite (hardness 3.5 to 4.5) and is brittle. The fracture is uneven and sometimes exhibits a distinct basal cleavage. The mineral

is opaque and shows a black streak. The specific gravity ranges from 4.58 to 4.65. Pyrrhotite is magnetic and is often referred to as magnetic pyrites. It is usually distinguished from other minerals by its color, hardness, and magnetic properties.

OCCURRENCE

Pyrite is widely distributed both geologically and geographically and occurs in rocks of all kinds and of all ages. It crystallizes within a wide range of temperatures, but the important deposits are mainly products of high or intermediate temperatures. It occurs abundantly in veins, disseminated deposits, and as masses along the contact of basic intrusives and sedimentary rocks. The deposits of economic importance usually occur as lenticular masses of great size. Deposits of pyrite may be of igneous, metamorphic or of sedimentary origin.

Outcrops of pyrite deposits are altered by oxidation and hydration to limonite and other iron oxides, and as pyrite is often associated with other valuable metallic minerals these iron-stained gossans frequently indicate valuable metalliferous deposits at depth.

The most important pyrite deposits occur in the province of Huelva, Spain, and the province of Alemtejo, Portugal. Other important deposits occur in Norway, Japan, Italy, Germany, France, and the United States.

Marcasite is much less abundant than pyrite but may occur with it. It is a relatively unstable mineral found mainly near the surface. It occurs in mineral veins as crystals, as nodules in chalk marl, and in coal and clay. It is also found in the lead-zinc ores of the Mississippi Valley.

Pyrrhotite is a high-temperature mineral and is not known to occur in ores of shallow or intermediate depths. It is found scattered in small quantities through igneous and thermometamorphic rocks and in mineral veins or lodes. Large massive deposits in gneiss and crystalline schists near their contact with granite are known. The sulphur content of pyrrhotite (38-39 per cent) is generally too low to be used as a source of sulphur unless impurities such as copper, nickel, or cobalt are present in sufficient quantities to render the deposit economic. Pyrrhotite ores containing copper are important sources of copper and sulphur in the Ducktown Basin of Tennessee. The pyrrhotite ores of Sudbury district in Ontario are mined chiefly for their copper and nickel content.

Analyses of pyrites from various places in the world are shown in the following tables.

Table 1.- Analyses of pyrites from various sources[a]
(Per cent)

	1	2	3	4	5	6	7	8	9	10	11	12
S	49.00	45.00	44.78	48.17	42.22	32.10	34.060	49.07	.50	46.50	48.5	38.72
Fe	43.55	42.50	37.49	42.32	36.30	35.94	53.150	44.28	-	44.25	-	35.94
Cu	3.20	3.50	-	.34	2.63	.30	.866	3.25	.70	1.00	2.1	2.24
Zn	.35	.25	4.23	-	2.35	4.68	-	-	-	-	-	4.50
Insol.	1.70	-	11.08	4.00	-	-	-	-	-	-	-	-
CaO	.14	.10	.87	-	0.52	1.00	-	.93	-	-	Tr.	[b]5.90
Mn	-	-	-	-	-	-	-	-	-	-	-	1.42
MgO	-	-	.20	-	0.29	.59	-	-	-	-	Tr.	[c]1.07
Sb	-	-	-	-	-	-	-	-	-	-	-	.08
As	.47	-	.07	Tr.	.023	.04	-	.38	-	.06	.01	.15
Pb	.93	.20	.14	-	-	.50	-	-	-	-	-	2.10
Au, oz.	-	-	-	.03	-	-	-	Var.	-	-	.01	.00011
Ag, oz.	-	-	-	.44	-	-	-	Var.	-	-	.40	.005
SiO_2	-	-	-	-	15.00	13.42	2.99	2.59	1.00-2.00	8.00	1.75	2.95
Al_2O_3	-	-	-	-	0.69	-	-	-	-	-	1.25	1.57
Alkali	-	-	-	-	-	-	-	-	-	-	-	1.84
Bi	-	-	-	-	-	-	-	-	-	-	-	.12

a - Columns headed by figures indicate source of ore, as follows: b - $CaCo_3$. c - $MgCo_3$

1. Rio Tinto, Spain.
2. Sulitelma, Norway.
3. Meggen, Germany.
4. Britannia Mining & Smelting Co., Canada (pyrites concentrates).
5. Lokken mine, Norway.
6. Louisa County, Va.
7. Virginia (pyrrhotite).
8. San Domingo, Portugal.
9. Hornet mine, California.
10. Leona Heights mine, California.
11. Skouriotisso mine, Cyprus.
12. Rammelsberg, Germany.

Table 2.- Estimated reserves of pyrite in Spain in 1926, by mines

Mines	Reserves (metric tons)				Per cent sulphur
	Actual	Probable	Possible	Total	
Concepcion	1,400,000	1,000,000		2,400,000	43
San Platon	750,000	500,000	1,000,000	2,250,000	48 to 50
Esperanza	799,000			799,000	47 to 48
Cueva de la Mora	3,047,600	722,000		3,769,600	44 to 45
Carpio	335,000	300,000	2,500,000	3,135,000	48 to 50
San Telmo	607,000	704,000		1,311,000	45
Pena del Hierro	3,000,000	1,000,000	2,000,000	6,000,000	
Rio Tinto	151,306,050	70,000,000		221,306,050	
La Zarza	35,000,000	12,000,000	8,000,000	55,000,000	
Perrunal y Lomero-Poyatos	6,000,000	2,000,000	4,000,000	12,000,000	47 to 50
La Joya	400,000	600,000		1,000,000	49.5
Castillo de Buitron	1,150,000	2,000,000		3,150,000	43 to 49
Santa Rosa	400,000	100,000		500,000	40 to 45
Romanera	200,000	800,000	1,000,000	2,000,000	
California-Concordia (Badajoz)		400,000	600,000	1,000,000	
Sotiel Coronada	1,080,000	580,000		1,660,000	30 to 44
La Torera		1,500,000		1,500,000	43.5
Tharsis	56,500,000	42,500,000	33,750,000	132,750,000	
La Lapilla	500,000	1,500,000	1,500,000	3,500,000	
Herrerias	2,000,000	2,000,000	3,000,000	7,000,000	47 to 49
Cabezas del Parto	1,500,000	750,000	1,250,000	3,500,000	
Fronteriza-Vuelta Folsa	200,000	210,000	400,000	810,000	
Nuestra Senora de Carmen	40,000	100,000	500,000	640,000	
Castilla de las Guardas	3,500,000	5,000,000	2,000,000	10,500,000	40 to 42
Silillos y Cuchichon (Sevilla)	1,671,633		3,500,000	5,171,633	42 to 47
Segunda Preciosa (Sevilla)	200,000	650,000		850,000	38
Caridad (Sevilla)	1,144,512	645,838		1,790,350	43
Total	272,730,795	147,561,838	65,000,000	485,292,633	

WORLD SOURCES

Spain

Spain has been the principal source of pyrite for many years, and in 1929 it produced about 50 per cent of the world output. The principal producing district is the Huelva district, which is located in an area about 100 miles long and 12 to 18 miles wide extending approximately east and west across the central part of the province of Huelva into the province of Alemtejo, Portugal. This area lies in the Sierra Morena Mountains and is located about 30 miles north of the port of Huelva, from which a large part of the district's output is exported.

The deposits consist of lenses of massive pyrite which occur in slates and porphyry and along the contacts of slate with porphyry or diabase. The lenses vary in size up to 6,000 feet in length, 800 feet in width and over 1,800 feet in depth. The geologic origin of these deposits has been a subject of great controversy for many years, and several theories of origin have been advanced. At present the hydrothermal replacement theory appears to be the most favored. The large pyrite masses are capped with gossans which usually extend more than 200 feet below the surface. The massive pyrite contains varying amounts of chalcopyrite and chalcocite, and some of the richer lenses have made the Huelva district one of the important copper producing districts of the world. The pyrite also contains small quantities of sphalerite, galena, and arsenopyrite. The average sulphur content of the ore shipped from the Huelva district is about 48 per cent. The average copper content is less than 1 per cent and is gradually decreasing.

The reserves of the Huelva district are sufficient for over 100 years operation at the present rate of production. Table 2 gives the reserves of pyrite in Spain by mines as of 1926:[3]

The principal mines are operated by two English companies, the Rio Tinto Co. (Ltd.) and the Tharsis Sulphur and Copper Co. (Ltd.). The former operates the Rio Tinto mines and the latter the Tharsis and La Zarza mines. These mines produce a large part of the total Spanish output and in 1926 contained 84 per cent of the total estimated pyrite reserves of Spain. Brief descriptions of the operations of the two companies follow.

Rio Tinto Co. (Ltd.) The mines of the Rio Tinto company are located at Rio Tinto, about 40 miles northeast of the port of Huelva. The deposits were worked intermittently for precious metals, lead, and copper from several centuries B. C. until 1873 when they were acquired by the present company; since then operations have been continuous and on a constantly increasing scale.

For many years copper was the principal source of revenue, but in recent years pyrite has become the principal product, notwithstanding increased copper production.

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3 - Rubio, -, and Mendizobal, -, Les Reserves Mondiales on Pyrites: Du XIV Congr s G ologique International Espagne vol. 2, Madrid, 1927, p. 464.

About 2,400,000[4] tons of ore is produced annually from large opencuts and underground stopes. The ore is dropped through chutes to haulage tunnels below the present working levels and is then delivered by trains to the various surface plants. The copper content of the ore varies considerably, but the average is under 2 per cent and is gradually decreasing. The ore is graded according to its copper and sulphur content. Ore containing about 3 per cent copper is sent directly to the smelter for the recovery of the copper. The sulphur content is dissipated into the smelter fume. Ore containing from 0.5 to 2 per cent copper is piled in large heaps on the surrounding hillsides and for several years is subjected to a leaching treatment which removes the copper and other soluble impurities and thus increases the sulphur content of the washed ore. The copper in the leaching solutions is precipitated on iron and recovered as cement copper. Ore with high sulphur content and less than 2 per cent copper is marketed directly either as straight sulphur ore containing about 0.3 per cent copper or as cupriferous pyrite with about 1 per cent copper.

The following table summarizes the operations of the company at Rio Tinto in 1929 according to official Spanish statistics.[5]

Table 3.- Operations of the Rio Tinto Co. in 1929

	Metric tons	Sulphur content, per cent	Copper content, per cent
Ore produced:			
Direct smelting ore	288,205	33.44	3.11
Leaching ore	517,757	47.46	1.17
Do., quartzose	708,583	28.99	1.55
Shipping ores	947,650	48.25	1.69
Total	2,462,195	-	-
Leaching operations:			
Ore in heaps, Dec. 31, 1929	21,943,446	-	-
Deposited in heaps during 1929	1,270,629	-	-
Iron consumed in 1929	25,714	-	-
Washed ore removed	521,471	47.46	-
Cement copper produced for market	11,413	-	92.46
Crude cement copper to smelter	8,664	-	64.86
Smelting operations:			
Materials charged:			
Rich mineral (sulphide)	251,383	-	3.46
Siliceous copper ore	72,759	-	2.57
Coarse cement copper	8,652	-	64.86
Fluxes (copper-free)	76,971	-	-
Coke	17,070	-	-
Blister copper produced	14,729	-	98.99

4 - American Motal Market, Estimates Rio Tinto Copper Production at 140,000,000 lbs. a Year: Vol. 38. No. 62, p. 2, April 1, 1931.

5 - Data taken from Consejo De Mineria. Estadistica Minera de Espana: 1929, p. 344-345.

The pyrite reserves of the mines of Rio Tinto as of 1926 were estimated at 221,-000,000 metric tons, 46 per cent of the total Spanish reserve and sufficient for nearly 100 years operation at the present rate of production.

Tharsis Sulphur and Copper Co. (Ltd.).- The Tharsis Sulphur and Copper Co. (Ltd.) has been operating in the Huelva district since 1866. The principal mines are La Zarza (Calanas) and Tharsis, both of which are connected by rail with the port of Huelva. La Zarza lies about 15 miles due west of Rio Tinto, whereas the Tharsis deposit is located about 15 miles southwest of La Zarza and 25 miles north of Huelva. These deposits are similar geologically to those at Rio Tinto, but the copper content is too low to justify leaching. According to official Spanish statistics[6] the company produced 672,000 metric tons of ore in 1928, which averaged about 0.6 per cent copper and 48.5 per cent sulphur. Of this output La Zarza contributed 586,000 tons and Tharsis 86,000 tons.

In 1926, the Tharsis and La Zarza mines were estimated to have reserves of 132,750,000 tons and 55,000,000 tons, respectively.

Norway

The pyrite and cupreous pyrite ores of Norway are a very important factor in the mining industry of that country. Some of the ores were mined for copper as early as the first half of the seventeenth century, but the first attempt to utilize the sulphur was in 1841 when small quantities were mined for a sulphuric acid factory near Trondhjem.

The principal pyrite deposits are distributed along the mountain chain which forms the backbone of Norway and are concentrated largely in three main districts; the Hardanger-Karmo district, the Trondhjem district, and the Grong district. Other deposits of minor importance occur along the west coast. Norway ranks second to Spain in pyrite reserves in Europe and second to Spain in world production.

The ore which averages about 44 per cent sulphur and 2.5 per cent copper is mined by underground methods. The Orkla-Grubeaktiebolag is the largest producing company, accounting for more than half of the total production. The Orkla is its largest and best-equipped mine.

There is only a small domestic consumption of pyrite in Norway, most of the material being exported, largely to Germany.

Italy

The mining of pyrites is of great significance to the national economy of Italy as a source not only of sulphur but also of iron. Processes have been developed for the recovery of iron from pyrite cinder to supplement the meagre iron-ore supplies of this nation.

Important deposits occur in the provinces of Grosseto and Torino, and deposits of less importance are situated in Genoa and Vicenza. Nearly all of the Italian production comes from the province of Grosseto in the department of Tuscany. The principal mines of

6 - Consejo De Mineria Estadistica Minera de Espana: 1928, p. 331.

Grosseto are situated at Gavorrano, Ravi, and Boccheggiano and are controlled largely by the Montecatini Corporation.

The production of pyrites in 1929 amounted to 698,550 metric tons, of which the Montecatini mines produced 534,269 tons. These figures include both iron and copper pyrites, the latter amounting to about 15 per cent of the total output. Italian pyrites are largely used in the manufacture of sulphuric acid, of which about 86 per cent is consumed by the superphosphate industry.

Japan

The mining of pyrites in Japan has been confined largely to this century, but due to increased demand for sulphuric acid, production has increased rapidly to nearly 600,000 long tons in 1928.

Deposits of pyrite are known on Shikoku Island, Honshu Island, and Kyushu. Pyrite occurs in bedded deposits, massive deposits, and as fissure veins, the bedded deposits being by far the most important. The ore is rarely free from copper and in recent years processes have been developed for extracting the copper by a chloridizing roast and subsequent lixiviation. The sulphur is recovered from the pyrites by ordinary roasting to SO_2. Cobalt oxide is recovered from the waste liquor, and the residue is used as a raw material for the iron works.

The Yanahara mine operated by the Fujita Mining Co. is the most important pyrite mine in Japan. It is located in the western part of Honshu Island. The ore from this mine contains about 50 per cent sulphur; this is somewhat higher than the average Japanese ore, which usually contains 40 to 45 per cent.

Germany

The larger part of the pyrites mined in Germany comes from the deposits at Meggen in Westphalia. The orebody consists of high-grade pyrite associated with barytes and a little galena, zinc blende, and chalcopyrite. The deposit has a thickness of 13 feet and has been followed along the strike for a distance of about 1½ miles. The ore runs about 75 per cent iron pyrite (FeS_2) and contains approximately 40 per cent of sulphur. Of minor importance is the Rammelsberg deposit, on the northern slope of the Harz Mountains, near the town of Goslar, which has been a producer of pyrites for many years.

France

France contains large and commercially important deposits of high-grade pyrites, located in the Departments of Rhone, Saone-et-Loire, and Gard.

In the Department of Rhone beds of pyrite exist on both banks of the Brevenne, a tributary of the Saone, for a width of 4 or 5 miles and are the largest producing deposits in France. The principal mines are the Mines de Sain-Bel which are located about 20 kilometers northwest of Lyon near Arbestle. To the north several kilometers is the old mine of Chessy. The pyrite of Sain-Bel and Chessy contains from 45 to 48 per cent of sulphur. Some of the pyrite contains copper up to 4 or 5 per cent but most of it is noncupreous.

The principal deposits in Saone-et-Loire are near Chalmoux where the Mines de Chizeuil are located. The average tenor of the ore is 42 per cent sulphur, with a minimum of 35 per cent.

Cyprus

An important deposit of pyrite is located near Skouriotissa in the district of Nicosia. The deposit is about 5 miles inland and is connected to the sea by a railroad. The ore contains about 48 per cent of sulphur and 2.5 per cent of copper. It is clean and uniform and has a low arsenic content (.01 per cent). No metallurgical work is done in Cyprus; the mine product is prepared for shipment in a crushing plant on the shore of Morphou Bay, near Pendayia.

Portugal

Pyrites is mined in Portugal along the southeastern boundary, on an extension of the Spanish pyrites belt. The largest producing mine (is San Domingos in the district of Beja, province of Alemtejo. The mineral occurs as lenticular masses in a mineralized zone 2,000 feet long and 200 feet thick. The ore is cupriferous and contains over 48 per cent sulphur and is similar to that mined at Rio Tinto. Portuguese pyrites is said to be of excellent grade. Other mines in the same district with ores of the same character are the Mina do Aljustrel and Mina Louzal.

Canada

Pyrites is available in many parts of Canada, but of late most of the production has been a by-product of copper, lead, and zinc mining, the pyrites being obtained in the process of concentrating the ores of these metals. The largest producer of this class of material is the Britannia Mining & Smelting Co., which produces about 70,000 tons of pyrite a year running about 49 per cent of sulphur. Other operations are being equipped to recover by-product pyrite in Canada. The bulk of the known deposits have been found during this century.

United States

The iron sulphides are widely distributed in the United States. They occur along the Appalachian Mountain region in lenses associated) with schists; in the Middle West associated with coal and lead and zinc deposits; in the Rocky Mountain States in numerous veins and lenses often mixed with the sulphides of other minerals; and in the coast ranges, usually in lenses more or less intimately connected with igneous rock.

No deposits of commercial importance are known or are likely to be found in Connecticut, Delaware, Florida, Iowa, Kansas, Louisiana, Michigan, Minnesota, Nebraska, North Dakota, Rhode Island, or Texas. All other States could furnish pyrites under suitable market conditions, although in Arkansas, Kentucky, Maine, Maryland, Massachusetts, Mississippi, New Hampshire, New Jersey, Oklahoma, and Vermont the probable reserves are small.

The location of the producing deposits in 1930 are shown in Figure 1.

California.- Production of pyrites in California in 1930 came from two mines: the Hornet mine near Keswick, Shasta County, and the Leona Heights mine near Oakland, Alameda County.

Figure 1.- Map of the United States, showing the location of the companies producing pyrites in 1930

Figure 2.- Map of the United States, showing the location of sulphuric-acid plants operating wholly or partly on pyrites

The ore from the Hornet mine is nearly pure iron pyrite containing 45 to 48 per cent of sulphur, not over 1 to 2 per cent of silica, and an average of 0.7 per cent of copper. The deposit occurs in a large fissure along a shear zone in rhyolite and has been developed for a length of several hundred feet. The pyrite is mined by underground methods and is crushed and screened to various specifications. After having produced some 200 to 300 tons daily for a period of 20 years, operations were suspended at this mine late in 1929 due to competitive pyrite sent into the San Francisco Bay region from the Britannia mine in British Columbia. The company, however, resumed full-scale operations during the third quarter of 1930.

The Leona Heights mine is operated by the Leona Chemical Co. and has a continuous record of production since 1913. The irregular massive deposits occur in a narrow belt of rhyolite that extends more or less continuously along the west front of hills bordering San Francisco Bay from Berkeley nearly to Decoto. The ore consists of pyrite mixed with chalcopyrite and pyrrhotite and a little silica. All of the production is now coming from a section of the deposit which has a thickness of 25 feet and is covered with 60 to 80 feet of overburden. The mine is developed by an adit. The pyrites averages about 47 per cent of sulphur and carries about $1 to $2 per ton in gold and approximately 1 per cent of copper, but the metal values are not recovered. The sulphur content is used by the Stauffer Chemical Co.

Tennessee.- Large quantities of sulphur-bearing ore are produced annually in the Ducktown district in Polk County. The ore from this region is largely pyrrhotite with small amounts of chalcopyrite, and is mined not only for its copper content but also for the sulphur which is used for the manufacture of sulphuric acid. Operations are conducted by the Tennessee Copper Co. and the Ducktown Chemical and Iron Co.

The Tennessee Copper Co. operates the Burra Burra mine and the Eureka mine. The Burra Burra mine is much the larger operation and is the source of the ore smelted and treated in the company's smelter and acid plant at Copperhill, Tenn. The mine operates steadily and furnishes a regular output of about 40,000 tons per month. The Eureka mine is operated irregularly and produces a much smaller tonnage of higher-sulphur ore, the rate of output depending upon the demand for sulphur.

During 1928 about 40 per cent of the combined output of these mines was treated by selective flotation and 60 per cent was smelted directly for copper. The pyrites, which is obtained as flotation concentrates, is separated into both pyrite and pyrrhotite.

The Ducktown Chemical and Iron Co. operates the Mary mine and the Isabella mine, but pyrites is recovered only at the Isabella mill where separate pyrite and pyrrhotite concentrates are made by selective flotation.

None of the pyrites produced in Tennessee enters the market, as both companies use all of their product in the manufacture of sulphuric acid.

Virginia.- Virginia contains a large deposit of pyrrhotite known as the "Great Gossan Lead," which lies in Floyd, Carroll, and Grayson Counties in the southwestern part of the State. The deposit was first mined for copper, and in the early fifties it was actively worked for its rich secondary copper ores. The deposit can be traced by its gossan outcrop for 18 miles and has an average width of about 35 feet. A complete chemical analysis of the

pyrrhotite shows that it contains 34.6 per cent of sulphur, 53.15 per cent of iron, 0.866 per cent of copper, and 2.99 per cent of silica. The Gossan mine of the General Chemical Co. at Cliffview, Carroll County, is near the southwest end of the "Great Gossan Lead" and is the only pyrites producer in Virginia at present. The ore is mined by opencut and underground methods and is used for the manufacture of sulphuric acid at Pulaski, Va. Formerly, the roasted material was sintered and used as iron ore in the blast furnace.

Wisconsin.- Pyrites is recovered in Wisconsin from ores produced in the Plattville district which contain principally sphalerite, pyrite, and galena. The ore is mined by underground methods similar to those used in the Southeast Missouri and Joplin districts. The mixed sphalerite-pyrite concentrates from the mills are brought together at the roasting plant of the National Zinc Separating Co. at Cuba City, Wis. A preliminary roast produces an oxide surface on the pyrite particles which makes them magnetic, and after the calcines are cooled, magnetic concentrates are made with a rotating magnetic disk. These magnetic concentrates are then given a further roast to remove more of the sulphur content. The gases from the preliminary and final roasts are used in the manufacture of sulphuric acid. Recently the separation of the pyrite and sphalerite by selective flotation has been attempted.

New York.- There is an occasional production of pyrite from the mines in the Edwards district, St. Lawrence County, N. Y. The most recent production has come from the Balmat mine near the town of Fowler. The production at present is a by-product from the flotation of zinc ores. In previous years the zinc and pyrite were separated by a magnetic process.

WORLD RESERVES

The world reserves of pyrites are large and widely distributed. Spain, Japan, Norway, Canada, and the Soviet Union are all plenteously supplied with this material. Definite figures concerning the pyrites reserves of the United States are not available, but the total amount that could be utilized is very large. In the Western States there are large reserves of pyrites which contain too little gold and silver to be mined at present, but which could be made available should conditions change. The application of selective flotation to base metal sulphide ores is making available quantities of pyrites which have been lost in the past. Pyrites also has been recovered commercially as coal brasses from the washing and cleaning of coal, and is being recovered from the sands at certain gold mines of the Rand, South Africa.

The world reserves of pyrites were studied by the Fourteenth International Geological Congress at Madrid in 1926, and the following table was compiled from data published by the Congress.[7] At the present rate of consumption the total reserves indicated in the table will supply the world with pyrites for many years. Spain has over 50 per cent of the total estimated supply.

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7 - Du XIV Congres Geologique International Espagne, Les Reserves Mondiales En Pyrites: 2 vols., Madrid, 1927, pp. 1-703.

Table 4.- Estimated World Reserves[1] of Pyrites as of 1926
(Metric tons)

	Actual reserves	Probable reserves	Possible reserves	Total reserves
Europe	355,782,795	234,139,558	97,336,000	687,258,353
Spain	272,730,795	147,561,838	65,000,000	485,292,633
Norway	40,473,000	15,715,000	Very considerable	56,188,000
Russia	9,338,500	25,077,000	1,550,000	35,965,500
Portugal	7,091,000	-	15,000,000	22,091,000
Italy	8,904,000	10,204,000	820,000	19,928,000
Germany	3,192,000	-	14,000,000	17,192,000
Sweden	1,581,000	14,870,000	-	16,451,000
Rumania	4,253,500	6,464,000	-	10,717,500
France	4,519,000	2,750,000	-	7,269,000
Greece	2,500,000	2,297,720	966,000	5,763,720
Cyprus	-	5,000,000	-	5,000,000
Austria	500,000	2,200,000	-	2,700,000
Finland	-	2,000,000	-	2,000,000
Poland	700,000	-	Small	700,000
Czechoslovakia	Considerable	-	-	-
Asia	101,533,192	58,041,135	7,500,000	167,074,327
Japan	93,338,192	56,651,135		149,989,327
Siberia	8,195,000	1,270,000	5,800,000	15,265,000
Netherlands East Indies	-	120,000	1,700,000	1,820,000
Africa	-	-	1,700,000	1,700,000
Madagascar	-	-	1,700,000	1,700,000
North America	1,500,000	32,000,000	10,000,000	43,500,000
Canada	1,500,000	32,000,000	10,000,000	43,500,000
South America	50,000	-	-	50,000
Argentina	50,000	-	-	50,000
Oceania	6,235,280	933,150	1,000,000	8,168,430
Tasmania	6,235,280	933,150	-	7,168,430
Queensland	-	-	1,000,000	1,000,000
Total	465,101,267	325,113,843	117,536,000	907,751,110

1 - Estimated reserves are necessarily understated because of no figures for the United States and other parts of the world. The totals for North America, South America, and Africa are especially incomplete.

PRODUCTION

Pyrites as mined is never chemically pure but contains admixtures of other base metal sulphides. Consequently, pyrites is produced either as a primary mine product or as a by-product in other mining operations, depending upon the kind and content of the base metal sulphides or other constituents. As a primary mine product, pyrites must necessarily come

from deposits capable of yielding a large tonnage of ore which is relatively free from worthless or deleterious materials and which can be cheaply mined and transported to market. Copper, zinc, calcium, magnesium, and sometimes lead are objectionable from the point of view of the acid maker, as they retain sulphur which oxidizes to sulphate and is not available for burning. The lead and zinc otherwise interfere by volatilizing. By-product pyrites is usually in the form of fine concentrates produced by selective flotation of base metal ores, although in the past some use has been made of the pyrite and marcasite removed in the treatment of coal. In Spain where the pyrites is cupriferous, the material is often leached after a period of atmospheric weathering. This method recovers a large percentage of the copper, and the residual ore known as washed pyrites is available for acid making.

The widespread occurrence of pyrite geographically and geologically allows considerable latitude in the production of this commodity. Europe, Asia, Africa, North America, and Australia all have produced significant amounts. Table 5 shows the production of pyrites in the various countries for the decade ending 1929 and for the year 1913.

Spain is by far the largest producer, contributing over 48 per cent of the world's production in 1929. Norway, the second largest producer, (9 per cent of 1929 production) has increased its production over 220 per cent since 1921 and 68 per cent over 1913. The Italian production, which was third in 1929 with a little less than 9 per cent of the total, has shown a strong tendency to increase. Even more pronounced is the trend in Japan, which produced 618,743 tons in 1929, or slightly less than 8 per cent of the total. No other country produced more than 5 per cent, yet this miscellaneous production amounted to over 26 per cent of the total.

Table 6 shows the estimated sulphur content of the world production of pyrite for the decade ending 1929 and for the year 1913. Since 1913 the total sulphur content of the pyrites has increased nearly 16 per cent, but during the four years preceding 1929 the figure was fairly constant.

Table 5.- World production of pyrite for 1913 and for the decade 1920 to 1929

(Metric tons)

Country	1913	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929
Algeria	-	8,294	18,891	7,446	21,000	17,870	12,566	11,505	12,918	13,825	16,804
Australia	10,380	10,628	6,832	11,905	12,073	-	-	-	-	-	-
Austria	[1]3,586	22,764	23,142	19,400	15,136	28,046	25,072	22,293	19,328	10,000	-
Canada	143,848	158,524	30,271	16,459	25,937	21,366	14,157	16,189	46,142	62,447	70,087
Cyprus [2]	-	1,422	11,737	29,667	62,509	141,050	176,036	152,776	211,462	243,913	295,772
Czechoslovakia	[3]62,750	44,786	35,147	8,968	-	15,935	21,594	22,512	23,300	23,626	23,005
France	311,167	140,375	188,640	186,790	201,715	194,157	212,559	192,340	203,700	202,084	194,430
Germany	268,583	436,271	414,437	328,724	193,028	159,623	223,293	237,870	350,430	342,179	351,909
Greece	128,867	3,239	49,625	54,510	52,290	76,262	65,000	81,000	100,050	94,270	[2]96,911
Hungary	[4] -	-	-	-	-	-	138	176	3,666	4,222	1,023
Italy	317,334	321,589	447,899	486,000	493,271	515,781	533,737	594,479	625,338	558,390	698,550
Japan	114,576	138,409	94,987	161,503	226,067	220,456	312,627	417,513	506,089	593,972	618,743
Norway	441,291	333,011	231,123	396,411	375,161	403,411	624,375	634,836	617,044	738,535	739,597
Poland	-	264	3,966	5,518	5,708	7,843	11,217	18,850	24,596	10,668	9,410
Portugal	435,812	247,972	239,771	199,046	203,555	216,805	217,296	210,487	301,035	242,122	384,349
Rumania	[5]50,854	10,889	15,937	20,381	25,512	32,657	26,983	42,039	25,514	23,715	23,851
Russia	77,513	[6]3,748	[6]11,010	[6]6,319	[6]22,167	[6]27,184	[6]54,996	[6]109,394	[6]225,318	[6]100,137	100,000
Spain	3,192,465	1,355,047	2,605,627	2,339,589	2,652,541	2,212,365	3,359,240	3,654,789	3,610,694	3,624,819	3,867,250
Sweden	34,319	107,326	45,772	57,321	58,297	66,353	69,873	69,759	69,239	19,996	72,055
United Kingdom	11,610	6,766	4,006	5,760	7,019	5,658	5,373	4,307	4,968	4,440	4,441
Union of South Africa	-	3,217	3,955	2,848	2,966	2,349	2,512	2,448	2,254	3,754	4,116
United States	346,816	315,765	159,640	175,625	193,694	170,609	196,749	226,933	307,685	317,836	338,817
Yugoslavia	[7]7,699	3,682	27,900	11,019	25,834	26,306	38,988	53,376	57,003	64,273	61,153
Total	5,959,470	3,673,988	4,670,315	4,531,209	4,875,480	4,562,086	6,204,381	6,775,871	7,347,773	7,299,223	7,972,273

1 - Figures do not include output of provinces subsequently incorporated in Czechoslovakia, Yugoslavia, and Rumania, which is shown under those countries.

2 - Exports.

3 - Includes production from provinces in Austria and Hungary subsequently incorporated in Czechoslovakia, as follows: Bohemia, 1,032 tons; Silesia, 510 tons; Hungary (Iglo), 61,667 tons.

4 - Production in 1913 was from districts subsequently incorporated in Czechoslovakia and Rumania and is shown under those countries.

5 - Production is from districts in Austria and Hungary which subsequent to 1913 were incorporated in Rumania.

6 - Data for year ended September 30.

7 - Output of Bosnia-Herzegovina only.

8 - Serbia only.

Table 6.- Estimated sulphur content of world's pyrite for 1913 and for the decade 1920 to 1929

(Metric tons)

Country	1913	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929
Algeria	-	3,898	8,900	3,350	9,450	8,400	5,950	5,386	6,071	6,498	7,730
Australia	4,000	4,000	2,700	4,700	4,800	-	-	-	-	-	-
Austria	[1] 1,000	3,301	3,700	3,424	3,163	6,031	5,712	5,137	3,153	2,000	-
Canada	57,000	61,333	11,079	6,260	10,045	8,838	6,883	8,142	22,887	35,007	34,000
Cyprus [2]	-	711	5,869	14,833	31,255	70,525	88,018	76,387	105,731	121,956	147,886
Czechoslovakia	[3] 25,000	18,000	12,000	3,600	-	6,400	8,854	9,005	9,203	9,332	9,200
France	150,000	67,000	88,213	85,963	91,250	88,432	96,587	90,748	94,380	97,000	93,000
Germany	95,391	164,749	151,202	118,053	62,767	48,144	82,041	98,939	149,531	145,866	149,983
Greece	62,000	1,571	24,068	26,437	24,277	35,401	31,525	39,577	48,884	45,360	[2] 46,500
Hungary	[4] -	-	-	-	-	-	60	70	1,500	1,700	400
Italy	149,000	151,846	206,277	207,236	227,382	237,398	243,922	273,248	286,227	256,683	320,000
Japan	46,000	55,000	46,000	65,000	90,000	88,000	125,000	167,000	202,000	237,000	250,000
Norway	195,335	141,567	122,919	170,241	161,721	173,389	266,903	272,330	266,855	321,630	323,844
Poland	-	100	1,600	2,600	2,800	3,100	4,500	7,500	9,800	4,300	3,800
Portugal	210,000	119,000	115,000	96,000	98,000	104,000	104,000	101,000	145,000	116,000	84,000
Rumania	[5] 23,000	4,600	6,700	8,600	10,700	13,700	11,300	17,600	11,000	10,000	10,000
Russia	31,000	[6] 1,500	[6] 4,400	[6] 2,500	[6] 9,000	[6] 11,000	[6] 16,800	[6] 44,000	[6] 90,000	[6] 41,000	40,000
Spain [9]	1,496,000	630,665	1,190,784	1,072,017	1,216,052	1,027,600	1,539,846	1,658,391	1,609,776	1,439,747	1,425,000
Sweden	12,698	39,603	17,867	22,105	22,479	25,616	26,695	27,073	32,126	12,394	32,082
United Kingdom	4,600	2,700	1,600	2,300	2,800	2,300	2,100	1,700	2,000	1,800	1,800
Union of South Africa	-	1,400	1,800	1,200	1,300	1,100	1,100	1,100	1,000	1,700	1,900
United States	139,000	125,958	68,201	73,736	77,465	68,490	76,998	87,042	115,226	115,124	122,303
Yugoslavia	[7] 3,500	1,700	13,000	5,000	12,000	[8] 12,000	18,000	24,000	26,000	29,000	27,000
Total	2,704,524	1,600,202	2,103,879	1,995,155	2,168,706	2,039,864	2,762,794	3,015,375	3,238,350	3,051,097	3,130,428

1 - Figures do not include output of provinces subsequently incorporated in Czechoslovakia, Yugoslavia, and Rumania, which is shown under those countries.

2 - Exports.

3 - Includes production from provinces in Austria and Hungary subsequently incorporated in Czechoslovakia.

4 - Production in 1913 was from districts subsequently incorporated in Czechoslovakia and Rumania and is shown under those countries.

5 - Production is from districts in Austria and Hungary which subsequent to 1913 were incorporated in Rumania.

6 - Data for year ended September 30.

7 - Output of Bosnia-Herzegovina only.

8 - Serbia only.

9 - Includes cupriferous pyrite, some of which has relatively low sulphur content which is not recovered.

USES AND SUBSTITUTES

The principal uses for pyrites are in the manufacture of sulphuric acid and sulphite wood pulp, for which purposes it is burned or roasted to furnish sulphur dioxide, an essential raw material in these industries. This operation generally takes place in kilns and furnaces of various types usually classed as lump burners and fines burners. In the mining and crushing of pyrites a considerable quantity of "fines" or "smalls" is made which formerly were unmarketable owing to the difficulty with which they were roasted. However, the introduction of efficient mechanical burners for roasting this fine material made available large quantities of ore which otherwise would have been of no economic value, such as pyrites ore, which has a tendency to break fine, and the pyrites concentrates recovered by selective flotation from plants treating base metal sulphide ores.

The lump burners or kilns are similar to simple deep bed coal fireplaces. They are rectangular in horizontal section, the grate area being about 5 feet wide and 4 to 6 feet long. The kilns are usually erected in batteries of 16 to 32 furnaces, the units being arranged in two rows, back to back, with a common central wall.

The fines burners are usually cylindrical multiple-hearth furnaces of the McDougal type with various modifications such as the water-cooled Wedge furnace and the Herreshoff furnace.

Recently there has been developed in Canada a new process for roasting fine pyrites in which dry air-floated material is kept in suspension in a vertical combustion chamber until the iron and sulphur have been oxidized. The resulting iron oxide settles to the bottom of the chamber and the gaseous sulphur products rise to the top where they are withdrawn for further treatment. The process is fully described by Freeman.[8] Similar operations are being conducted experimentally at several plants in the United States.

The iron-oxide residue from pyrites burners can be converted into a suitable raw material for the iron blast furnace by sintering. The sulphur-dioxide gas is usually treated for the removal of suspended dust particles before it is used in the manufacture of sulphuric acid or sulphite wood pulp.

Pyrite has long been a competitor of native sulphur in the manufacture of acid and sulphite wood pulp. At present there is a distinct preference for native sulphur because of the ease of obtaining a constant flow of clean sulphur-dioxide gas, and, where necessary, of obtaining higher concentrations of the gas. It is necessary to handle and transport more than 2 tons of pyrite in order to obtain the equivalent sulphur content of 1 ton of native sulphur. The technical improvements made in burners of fine pyrites has helped the position of pyrites some, but where fines burners are employed, a dust chamber to arrest the dust carried forward from the furnaces is necessary. With large reserves of native sulphur available in the United States coupled with the simplicity of burning native sulphur it is not surprising that pyrites is receiving a diminishing percentage of the sulphur market. The use of pyrites now depends entirely on a very favorable price differential per unit of sulphur.

At present only about 20 per cent of the sulphuric acid produced in the United States is derived from pyrites, whereas in Europe sulphuric acid is largely made from pyrites

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8 - Freeman, Horace, The Utilization of Pyrites in Pulp in Acid Manufacture: Canadian Min. and Met. Soc., Bull. 216, Apr., 1930, pp. 471-476.

in Great Britain and Germany appreciable quantities are made from sulphur. European producers of pyrites are developing processes for the production of elemental sulphur from pyrites.

Pyrite is also used in smelters as a fluxing agent to furnish iron for the slag. In pyritic and semipyritic smelting, the oxidation of pyrite provides some heat for the smelting operation, and the resultant sulphur-dioxide gas is sometimes used in the manufacture of sulphuric acid. Pyrite containing gold and silver is frequently treated in copper and lead smelters, where its precious metal content is recovered in the metal bullion. The pyrites used in smelting operations seldom is obtained in the open market.

Minor uses of pyrites are in radios, jewelry, vermillion paints or for the manufacture of copperas, which is used in dyes, writing ink, wood preservatives, disinfectant, and certain kinds of fertilizers. In late historic times it was used in some of the old wheellock guns.

Figure 2 shows the location of sulphuric-acid plants in the United States which use pyrite as a raw material.

CONSUMPTION

The first application of pyrites in the making of sulphuric acid was described in an English patent in 1818, but there was not much development along this line until 1838, when the Sicilian Government granted a monopoly for the exportation of Sicilian sulphur to the Marseilles firm of Taix & Cie, which immediately tripled the price of sulphur. Search for substitutes for sulphur in the manufacture of sulphuric acid was instituted, and in 1839 pyrites was first used on a large scale in England for this purpose. In the succeeding years pyrites gradually but steadily replaced sulphur for the production of sulphuric acid, notwithstanding the fact that the Sicilian monoply of sulphur did not endure and the price was returned to normal.

In 1891 the sulphur content of the pyrite consumed in the United States was 93,233 long tons, which was obtained from domestic ores and foreign ores in about equal amounts. At the beginning of the century (1900) this had increased to 237,195 tons, with 39 per cent being furnished by domestic ores. The amount of pyrites consumed in the United States continued to increase until in 1916 when the consumption of pyrites - that is, domestic production plus imports - amounted to about 740,000 tons, with over 75 per cent of the ore coming from abroad. Meanwhile extensive reserves of cheaply mined sulphur had been developed in the United States.

The advent of the World War had a tremendous effect on the consumption of pyrite in the United States. The need of transportation facilities for essential traffic between the United States and Europe resulted in the drastic curtailment of bottoms engaged in other traffic, and imports of pyrites from Spain and Portugal fell off materially. American manufacturers of sulphuric acid then turned to brimstone as a source of sulphur. At the close of the war many industries, because of an assured domestic supply, continued to use brimstone rather than change their plants to handle pyrites. Thus the consumption of pyrites in the United States has been impeded by an increasing application of sulphur in the manufacture of acid.

Pyrites is the principal source of sulphur for the manufacture of sulphuric acid in Europe, where approximately 50 per cent of the world's output of the acid is made. Considerable acid in Europe is made from zinc blende and smaller amounts from spent oxide and native sulphur. Recently gypsum has been used as a raw material in acid manufacture but as yet is of minor importance.

In 1900, the world's consumption of sulphur amounted to 2,500,000 metric tons, with about 80 per cent coming from pyrites. In 1929 the world's consumption had risen to approximately 6,000,000 tons, derived almost equally between pyrite and sulphur. The pyrites producers have been forced to a lower unit price of sulphur than the unit price of sulphur in native sulphur in an endeavor to retain their share of the markets. Another important competing factor is the very large quantity of sulphuric acid now manufactured at zinc smelters and to a lesser extent at copper smelters. With the present tendency away from pyrites to sulphur, methods are being perfected for the production of sulphur from pyrites. Plants are now being constructed in Norway and Spain for this purpose.

MARKET AND PRICE

The principal market for pyrites in the United States is located along the Atlantic seaboard where Spanish pyrites, mined cheaply and favored by cheap ocean transportation, can compete with native sulphur in the fertilizer industry. The chief marketing point is New York, but there are subagents at various points along the Atlantic seaboard. As it is necessary to handle over 2 tons of pyrite to obtain a sulphur content equal to 1 ton of native sulphur, the cost of transportation is a controlling factor in the marketing of pyrites. Consequently, only pyrites produced in close proximity to markets for sulphuric acid can be marketed profitably. Most of the domestic pyrites is produced by consuming companies, so that only a small part enters the open market. Pyrites produced in California is marketed in the San Francisco Bay region. Some Canadian pyrites is marketed in the Eastern States.

Pyrites is usually sold at a price per unit of sulphur, and a specified sulphur content is usually guaranteed. The unit is 1 per cent per ton of 2,240 pounds or 22.4 pounds of sulphur. Two grades are recognized, lump and fines. Lump ore ranges in size from 2 to 10 inches, and the proportion of fines must not exceed 10 per cent past a 1/2 or 3/8 inch screen All material past 1/2 or 3/8 inch screen is sold as fines, which usually sell for 2 cents less per unit of sulphur than lump ore. Sometimes run-of-mine ore is sold, but this is objectionable, as it usually carries a large proportion of fines.

No well-defined premiums or penalties are recognized by the trade, but where transactions involve uncertainty, special clauses are inserted in the sales contract. A premium is usually allowed where the sulphur content exceeds the guarantee basis and a penalty where it is less. No allowance is made for moisture except where it is in excess of 1 per cent. Impurities are present, usually in small amounts; zinc and lead where percentages approximate 3 per cent or more are objectionable; arsenic in excess of 1 per cent is considered objectionable in acid manufacture, as it renders the acid unfit for certain uses and has a deleterious effect on some catalysts used in the contact process. Pyrites containing recoverable amounts of copper may be sold outright to the buyer who sells the residual iron oxide after the copper has been removed by leaching, or it may be leased to a buyer who pays only a specified price for the sulphur content, the seller retaining ownership of the sinter.

The following table shows a typical analysis of imported Spanish pyrites which is marketed on our eastern seaboard:

Table 7.- Typical analyses of Spanish pyrites imported into the United States[1]

	Pyrites lumps, per cent	Pyrites fines, per cent	Washed pyrites, per cent
Copper	1.75- 2.00	1.75- 2.00	0.30 - 0.45
Lead	.94	1.10	.50 - .80
Bismuth	0.14	.017	.005 - .01
Arsenic	.45	.47	.25 - .35
Antimony	.055	.062	.010 - .020
Sulphur	47.47	46.43	48.50 - 50.00
Phosphorus	.007	.008	.007 - .010
Iron	40.41	39.27	42.0 - 43.50
Alumina	1.69	1.25	.10 - .20
Zinc	1.31	1.37	.35 - .55
Manganese	.041	0.60	.01 - .02
Nickel and cobalt	.122	.134	.13 - .132
Lime	.324	.370	.20 - .30
Magnesia	.115	.130	.10 - .15
Siliceous residue	3.59	4.90	2.50 - 4.00
Silver	.0047	.0050	.004 - .005
Gold	.0007	.00007	.0007 - .00007

1 - Hearings before a subcommittee of the Committee on Finance, United States Senate, Seventy-first Congress, 1929, p. 663.

The price of Spanish pyrites per long-ton unit, c.i.f. Atlantic ports from 1926 to 1930, inclusive, is shown in the following table:

Table 8.- Prices of Spanish pyrites per long-ton unit, c.i.f. Atlantic ports, from 1926 to 1930, inclusive[1]

Year	January 1	April 1	July 1	October 1
1926	$0.11½-.12	$0.12-.13	$0.12-.13	$0.12-13
1927	.12 -.13	.13-.13¾	.13-.13¾	.13-13¾
1928	.13 -.13¾	.13-.13¾	.13-.13¾	.13-13¾
1929	.13 -.13¾	.13-.13¾	.13-.13¾	.13-13¾
1930	.13 -.13¾	.13-.13¾	.13-.13¾	.13-13¾

1 - Data taken from Oil, Paint and Drug Reporter.

IMPORTS AND EXPORTS

The United States imports large quantities of pyrites. Over half of the domestic consumption is met by foreign material. Table 9 shows the imports of pyrite by countries of origin during the last decade as compared with those of 1913. While there was a noticeable drop in the imports of pyrites immediately following the war, there has been a large increase since 1924, but at present imports of pyrites are only 50 per cent of the pre-war figure. Table 10 shows the distribution of imports by customs districts for the decade ending 1929 and for 1913.

Table 9.- Imports of pyrites into the United States by countries of origin, 1913 and 1921 to 1930

(Long tons)

Country	1913	1921	1922	1923	1924	1925	1926	1927	1928	1929	1930
Canada	31,293	7,000	66	10,443	3,935	1,211	1,046	9,203	56,956	68,243	42,117
Cuba	-	16,250	1,000	-	-	-	-	-	-	-	-
Portugal	118,732	-	-	-	-	-	-	-	-	-	-
Spain	814,534	191,515	266,872	252,862	242,786	275,174	365,105	241,591	393,840	446,093	325,992
Other countries	-	1,464	7,516	390	16	-	-	-	7,468	-	
Total	964,559	216,229	275,454	263,695	246,737	276,385	366,151	250,794	458,264	514,336	368,114

Table 10.- Imports of pyrites into the United States by customs districts, 1913 and 1921 to 1930

(Long tons)

Customs districts	1913	1921	1922	1923	1924	1925	1926	1927	1928	1929	1930
Buffalo	-	-	40	10,401	233	995	79	174	120	413	90
Chicago	-	7,000	-	-	-	-	-	-	28	-	-
Georgia	114,854	3,566	-	11,125	9,451	7,713	7,179	3,650	5,915	-	5,554
Jacksonville	20,093	-	-	-	-	-	-	-	-	-	-
Maine and New Hampshire	-	-	-	-	-	-	-	-	-	25,751	-
Maryland	185,546	67,539	93,572	81,094	89,741	94,331	128,128	74,827	140,090	182,249	175,611
Massachusetts	30,463	-	-	-	-	-	-	-	14,907	-	-
Mobile	42,795	-	-	-	-	-	-	-	-	-	-
New Orleans	26,894	-	-	-	-	-	-	-	-	-	-
New York	128,823	26,680	29,572	46,078	38,411	24,016	29,693	25,895	70,231	54,331	42,145
North Carolina	37,311	-	1,000	-	-	-	-	-	-	-	-
Ohio	-	-	1	-	3,500	36	-	-	-	-	-
Pensacola	19,902	-	-	-	-	-	-	-	-	-	-
Perth Amboy	69,434	-	-	-	-	-	-	-	-	-	-
Philadelphia	147,258	104,330	143,165	94,549	87,122	128,568	174,476	120,309	153,644	166,056	87,178
San Francisco	-	-	-	390	-	-	200	7,488	50,147	52,514	7,990
South Carolina	85,213	-	2,000	10,801	10,745	13,546	14,272	8,980	11,792	5,696	7,322
St. Lawrence	-	-	25	-	-	-	-	-	-	-	-
Vermont	27,244	-	-	-	2	23	-	1,280	6,554	17,326	-
Virginia	10,632	7,114	6,079	9,215	7,352	7,000	11,357	7,930	4,729	10,000	27,778
Washington	-	-	-	42	200	157	767	261	107	-	14,446
Undistributed	18,097	-	-	-	-	-	-	-	-	-	-
Total	964,559	216,229	275,454	263,695	246,757	276,385	366,151	250,794	458,264	514,336	368,114

Spain and Norway have the largest export trade in pyrites. Other countries notably Italy, France, and Germany have some export trade but also have a large import trade especially France and Germany. The production from Cyprus and Portugal is exported entirely to foreign markets. Tables 11 and 12 show exports of pyrite from Spain and Norway during the period 1925 to 1929.

Table 11.- Exports of pyrites from Spain by countries, 1925 to 1929

(Metric tons)

Country	1925		1926		1927		1928		1929	
	Non-cupreous	cupreous	Non-cupreous	cupreous	Non-cupreous	cupreous	Non-cupreous	cupreous	Non-cupreous	cupreous
Algeria	8,902	–	20,994	–	31,187	5,081	22,394	–	29,499	7,821
Argentina	–	–	–	–	–	10,705	12	–	–	–
Austria	–	11,242	–	2,843	–	–	–	–	–	–
Belgium	155,208	32,242	110,257	40,328	146,538	23,481	192,918	13,537	260,268	16
Denmarck	11,020	55,545	21,681	38,649	17,337	47,350	22,541	64,636	16,286	68,423
Finland	–	–	–	–	3,007	–	–	–	–	–
France	581,563	435	628,308	–	537,723	–	505,191	–	556,018	14
Germany	111,349	90,637	89,189	78,248	146,591	153,442	82,941	150,453	89,262	168,850
Great Britain	157,985	83,919	157,926	70,761	179,231	71,571	163,411	79,192	182,181	121,652
India	–	–	–	–	–	6,097	–	–	–	–
Italy	93,323	10,951	54,085	24,500	48,906	22,687	25,861	16,596	64,411	12,191
Morocco	5,614	–	4,655	–	6,446	–	7,491	–	7,397	201,033
Netherlands	245,108	227,117	245,309	184,675	418,881	267,072	508,842	228,931	459,649	–
Norway	7,641	–	–	–	–	–	–	–	–	1,581
Portugal	–	–	–	–	–	–	4,887	–	–	–
Sweden	–	4,136	–	2,432	–	14,257	–	8,972	18,636	13,189
U. S. A.	21,401	320,904	13,699	333,742	17,435	269,479	135,661	296,810	126,258	317,126
U. S. S. R.	–	–	4,516	–	–	–	–	–	–	–
Total	1,399,114	837,128	1,350,619	776,178	1,553,282	891,222	1,672,150	859,117	1,809,865	911,896

Table 12.- Exports of pyrites from Norway by countries, 1925 to 1929

(Metric tons)

Country	1925		1926		1927		1928		1929	
	Non-cupreous	Cupreous	Non-cupreous	Cupreous	Non-cupreous	Cupreous	Non-cupreous	Cupreous	Non-cupreous	Cupreous
Belgium	–	28,550	1,680	26,975	1,715	19,525	3,750	20,080	6,084	14,506
Denmark	1,986	25,699	3,090	36,324	1,505	6,461	–	4,028	–	3,772
Danzig	–	–	–	–	–	–	2,300	–	20,105	–
Finland	–	–	3,800	–	5,580	–	–	–	–	–
France	4,315	23,679	1,880	64,145	18,445	51,282	1,990	70,188	–	55,038
Germany	62,657	207,501	88,124	193,985	108,084	213,916	147,167	216,823	118,750	267,160
Great Britain	20,183	6,805	12,407	3,300	18,249	11,382	17,569	8,427	16,435	21,175
Lithuania	–	–	–	2,100	–	8,095	–	8,825	5,834	2,376
Netherlands	–	18,570	1,056	11,550	1,800	–	2,810	550	12,874	–
Poland	–	–	–	–	–	–	–	–	–	–
Russia	–	–	–	–	–	–	–	–	–	–
Sweden	1,912	136,814	17,694	114,662	23,524	113,594	32,806	108,904	24,164	88,259
Total	91,053	447,618	129,731	453,041	178,302	424,255	208,392	437,825	204,246	452,286

Table 13.- Exports of pyrites from Cyprus by countries of destination, 1924 to 1928
(Long tons)

Country	1924	1925	1926	1927	1928
Belgium	1,328	-	-	-	-
Egypt	-	-	3,500	-	-
France	-	3,101	3,200	6,800	2,400
Germany	49,041	67,880	59,872	52,250	42,635
Holland	7,500	30,123	17,454	34,110	57,099
Italy	56,396	62,955	56,337	101,604	100,826
Poland	-	-	-	-	18,300
United Kingdom	13,923	9,196	10,000	13,358	18,800
Total	128,188	173,255	150,363	208,122	240,060

Table 14.- Exports of pyrites from Portugal by countries of destination, 1923 to 1927
(Metric tons)

Country	1923	1924	1925	1926	1927
Belgium	33,328	37,627	17,274	21,796	40,426
England	36,991	23,519	18,189	13,139	16,447
France	73,943	100,968	108,097	88,544	84,904
Germany	21,680	3,364	113	-	4
Holland	13,085	16,500	2,921	-	2,596
Italy	-	-	2,429	11,161	15,499
Spain	1	-	-	-	-
Sweden	3,580	-	-	-	-
United States	-	-	-	1,747	674
Total	182,608	181,978	149,023	136,387	160,548

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I. C. 6524

September, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

UTILIZATION OF DOLOMITE AND HIGH-MAGNESIUM LIMESTONE

BY

Paul Hatmaker

I.C.6524.
September, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

UTILIZATION OF DOLOMITE AND HIGH-MAGNESIUM LIMESTONE[1]

By Paul Hatmaker[2]

CONTENTS

INTRODUCTION

This paper covers briefly the particular field of dolomite and magnesium limestone; certain uses of high-calcium limestone are mentioned only to show more clearly the limitations of the magnesian rocks. Such a comparison is helpful because magnesium carbonate is essential for some uses; for others it is immaterial; and for many its presence in an appreciable amount is objectionable.

Magnesium carbonate, occurring as a rock resembling limestone or dolomite, is described in United States Bureau of Mines Information Circular 6437, Magnesite. Other compounds of magnesium are described in United States Bureau of Mines Information Circular 6406, Magnesium Compounds (Other Than Magnesite).

Limestone, geologically, is a rock composed essentially of calcium carbonate; commercially, however, the term may include dolomite and magnesium limestone.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used. "Reprinted from U. S Bureau of Mines Information Circular 6524."

2 Mining engineer, building materials section, U. S. Bureau of Mines.

Dolomite is a distinct mineral composed of calcium and magnesium carbonates combined in molecular proportions as expressed by the formula $CaMg(CO_3)_2$. Dolomitic limestones are those which, though containing considerable dolomite, include in addition either calcium carbonate or a physical mixture (not a chemical combination) of calcium and magnesium carbonates. Ordinarily the term "limestone" means a rock containing less than 10 per cent of magnesium carbonate. With more than 10 per cent of magnesium carbonate the term "magnesium limestone" is used until the amount is about 40 to 45 per cent, when such rock is called "dolomite." The term "high-calcium limestone" commonly means one containing at least 95 per cent of calcium carbonate and not more than 5 per cent of magnesium carbonate. "Low-magnesium limestone" applies in a general way to rocks in which the magnesium carbonate content is between 10 and 20 per cent; with more than 20 per cent magnesium carbonate the term "high-magnesium limestone" may be employed. Many rocks containing a high percentage of magnesium carbonate are not true dolomites, but dolomitic or high-magnesium limestones. They are, however, commonly known as dolomite. No very definite lines of demarcation have been fixed to designate the various gradations from high-calcium limestone to pure dolomite.

Dolomite, at best, is a low-priced commodity. Extensive deposits are found in many localities, but for many of its uses it must compete with limestone, which is even more abundant and widely distributed. For some purposes it is even less desirable than limestone, and for those purposes for which its magnesium content is important it meets competition from other sources of magnesium, notably magnesite and natural magnesium salts and bitterns. Even sea water contains magnesium which is occasionally extracted therefrom, but frequently the liquors which remain after common salt (sodium chloride) has been extracted are allowed to go to waste, although they may contain considerable magnesium. In localities remote from manufacturing activity, dolomite deposits must be recognized as having little present commercial value and even in the industrial areas there is nothing that approaches an open market. Consumers, as a rule, either own quarries or have necessarily established contractual relations which guarantee their supplies.

Quarry operations are begun for a variety of reasons and the control of a favorable quarry site is one, and often only a minor factor in the successful establishment of a new enterprise. Before undertaking the working of a dolomite or magnesium limestone deposit the prospective operator should give particular thought to all the economic factors involved, among which may be mentioned (1) Markets and market demands, (2) adequare supply and uniformity of raw material, (3) capital requirements including working capital after the producing stage has been reached, (4) pr duction costs, and (5) possible competition.

DESCRIPTION

Dolomite occurs massive, finely or coarsely granular, or crystallized in small rhombohedral crystals which often have characteristically curved faces. The color varies from white to gray or brown; or more rarely to blue, pink, or yellow. Many magnesium limestones can be distinguished from calcium limestones only by chemical tests. Weathered dolomite is typically chalky white, but variations in magnesium content or impurities may cause differences in color and general appearance.

Composition

The theoretical composition of pure dolomite is 54.3 per cent calcium carbonate ($CaCO_3$), and 45.7 per cent magnesium carbonate ($MgCO_3$). Expressed in another way it contains

lime (CaO) 30.4 per cent, magnesia (MgO) 21.8 per cent, and carbon dioxide (CO_2) 47.8 per cent.

Magnesium limestones may vary in composition from practically pure $CaCO_3$ with only a few per cent of $MgCO_3$ to a dolomite. Very rarely the percentage of magnesium carbonate is greater than that of a dolomite and may grade to magnesite which, theoretically, is pure $MgCO_3$.

Properties

Dolomite (hardness 3.5 to 4) is slightly harder than calcite but generally softer than fluorite. The specific gravity is 2.8 to 2.9, therefore it is slightly heavier than. Calite (specific gravity 2.72). Pure dolomite does not melt; application of sufficient heat drives off carbon dioxide leaving calcium and magnesium oxides. The temperature of this dissociation is about 850°C. or 1,562°F. Crystals of dolomite are pearly or vitreous. If granular, however, the luster is dull or earthy. Dolomite crystals are brittle and break with an uneven to conchoidal fracture. Fragments transmit light poorly or not at all. Certain physical properties such as porosity and crushing strength vary among different deposits.

High-calcium limestones effervesce freely in cold dilute hydrochloric acid, but dolomite and high-magnesium limestones effervesce only if the acid is heated. If polished portions of magnesium limestone are etched by cold weak acid and then examined under a microscope the true dolomite will be unaltered, but the separate calcium carbonate will show surface effects of solution.

THE USES OF DOLOMITE AND HIGH-MAGNESIUM LIMESTONE

Those uses which to a varying degree are dependent upon or affected by the chemical composition of the stone are in (1) refractories, (2) "technical carbonate", (3) certain processes of paper manufacture, (4) some types of lime mortars, (5) blast-furnace flux, (6) Vienna lime, (7) glass works, (8) carbon dioxide and chemicals, (9) agriculture, (10) paints, kalsomine, whitewash, and varnish, (11) ceramics, (12) prepared whiting, (13) rubber preparations, (14) tanneries, (15) fungicides, and (16) such miscellaneous uses as in pig-iron casting, in "pickling" iron and steel, in wire drawing, and in the manufacture of strawboard.

Dolomite, like limestone, may be used as building stone, and as crushed stone, both of which uses, of course, are contingent mainly upon physical properties.

Uses for which Chemical Properties are Important

Refractories

Dolomite and high-magnesium limestone are used extensively as substitutes for magnesite refractories in basic open-hearth steel furnaces and basic Bessemer converters; in lead-refining reverberatory furnaces, lead cupelling furnaces, and crucibles for lead blast furnaces; in copper converters, and copper reverberatory furnaces; and in the form of crucibles for melting metals. Dead-burned material in various forms is commonly used, although raw dolomite may be employed for minor repair work.

Dead-burned dolomite is made by calcining dolomite or high-magnesium limestone to about 1,500°C. either in a blast furnace or in a special kiln. At this temperature virtually all the CO_2 is driven off, leaving CaO and MgO which sinter to an extent depending upon the impurities present. Many manufacturers add certain agents such as iron oxide, alumina, and silica to the crushed dolomite before calcining, which aids the sintering action. In one process these agents are ground with the dolomite in a wet mill. The slurry is subsequently calcined and clinkered in a rotary kiln similar to that used in the manufacture of cement.

Recently, some dolomite deposits have been found in Missouri which contain the necessary impurities in the right proportion. The use of this stone, dry crushed to granular size and dead-burned, has resulted in a production of a high-grade material, said to be superior to the synthetic mix.

There are two general ways of utilizing dolomite as refractory-masonry work; First, burned dolomite is mixed with tar or a fluxing agent and is applied as a monolithic lining; second, burned dolomite, to which is added tar or suitable fluxing agents, is formed into brick shapes which are fired and then laid in the same manner as other refractory brick.

In steel furnaces the sides and top are usually made of silica brick. Two or three courses of fire brick are laid in the bottom upon the steel shell. There are then added two or more courses of magnesite brick which are stepped up at the sides. The basic bottom is spread directly on the magnesite brick. This material usually is dead-burned grain magnesite sometimes mixed with hot tar or molasses. A binder of 5 to 20 per cent of basic-cinder slag is added. The grain magnesite is fused on in successive layers building up the front and back walls well above the slag line.

Where dead-burned dolomite is used as a substitute for magnesite in such basic bottoms, the raw stone should contain less than 1 per cent of SiO_2, less than 1.5 per cent of combined Al_2O_3 and Fe_2O_3, and at least 35 per cent of $MgCO_3$, the rest being calcium carbonate ($CaCO_3$). Dolomite is somewhat inferior to grain magnesite for this purpose; however, raw or calcined dolomite is satisfactory for minor repair work which requires 40 to 50 pounds or more per ton of steel. The following table compiled by the Bureau of Mines indicates the extent of the dead-burned dolomite industry.

Dead-burned dolomite sold or used by porducers in the United States, 1920-1929

Year	Reported as raw stone		Reported as dead-burned		Total quantity calculated as dead-burned, short tons
	Short tons	Value	Short tons	Value	
1920	612,800	$742,020	316,293	$3,732,522	623,000
1921	79,480	85,786	107,664	1,113,010	147,000
1922	146,640	172,251	348,838	2,813,946	422,200
1923	205,890	249,993	357,642	3,599,116	460,600
1924	309.010	287,137	328,659	3,209,257	483,200
1925	415,710	375,315	392,145	3,730,509	600,000
1926	511,300	458,802	386,715	3,593,731	642,000
1927	434,160	424,140	374,415	3,459,803	591,000
1928	522,850	509,502	448,761	4,283,036	710,000
1929	516,400	461,444	488,032	4,261,942	746,000

Technical Carbonate

"Technical carbonate" finds its widest utilization in the manufacture of pipe and boiler covering, and for general heat insulation, but it is also used in pharmacy, in the rubber trade, as an accelerator, and as a constituent of certain paints, varnishes, glass, printing inks, cosmetics, table salt, tooth paste, and other commodities. Other names for this material are basic magnesium carbonate, "block magnesia," and magnesia alba. Both in England and in the eastern United States technical carbonate is made chiefly from dolomite by the Pattinson process or a modification thereof. The process of manufacture from dolomite is as follows:

Dolomite mixed with coke is first calcined, thus driving off the carbon dioxide, which is recovered, purified, compressed, and cooled. The calcined rock, essentially a mixture of CaO and MgO, is slaked in water and then recarbonated with the recovered CO_2. In this operation there is made an insoluble calcium carbonate and a soluble bicarbonate of magnesia ($Mg(HCO_3)_2.H_2O$). The calcium carbonate is removed by filtration and the filtrate is boiled, which drives off some of the carbon dioxide and precipitates a white basic magnesium carbonate somewhat variable in composition but considered as having the formula

$4MgCO_3,Mg(OH)_2.5H_2O$.

For the manufacture of the so called "85 per cent magnesia," molded insulation, "technical carbonate" is mixed with asbestos fiber and perhaps other bonding agents, molded into the desired form, dried for 5 or 6 days, and finally cut to true dimension with special machinery.

The following table shows the amount of dolomite used in the manufacture of "technical carbonate."

Dolomite sold or used by producers in the United States for manufacture of basic magnesium Carbonate, 1920-1929

Year	Short tons	Value
1920	57,300	$107,107
1921	32,050	60.648
1922	53,170	79,313
1923	116,410	146,337
1924	98,160	129,390
1925	98,980	131,440
1926	72,850	110,560
1927	69,940	115,932
1928	94,200	122,260
1929	84,750	129,383

Paper Mills

One of the most important methods of making paper from wood is the sulphite process, which is used mainly for the wood of coniferous trees, such as pine, spruce, and hemlock. By digestion in an acid liquor under high temperature and pressure, all the constituents of the wood chips except cellulose are dissolved and removed. This acid liquor is a solution of magnesium and calcium bisulphites, together with more or less free sulphur dioxide, and is obtained by treating either milk of lime or wet limestone with sulphur dioxide, prepared by burning sulphur or iron pyrites in air.

Dolomitic or high-magnesium lime is preferred for preparing the acid liquor from milk of lime. Magnesium bisulphite is said to be more stable, more soluble, milder, and more effective in its chemical action than calcium bisulphite.[3] Its decomposition products are more soluble and it produces a softer and whiter pulp. Quicklime or hydrated lime used in preparing this liquor should have a minimum of 94 per cent of $CaO + MgO$, a maximum of 5 to 10 per cent of CO_2, and not more than 3 per cent of Fe_2O_3, Al_2O_3, SiO_2, and insoluble material.

The acid liquor is also prepared by the Jennsen tower system whereby sulphur gases pass up through a tower packed with limestone. There appears to be some objection to using a limestone high in magnesium, because such material is said to break down and clog the tower, thereby hindering gas absorption. According to Claudet[4] a limestone for the tower process should preferably have not more than 2½ per cent of $MgCO_3$, although in some cases 3 per cent may be tolerated and rarely 5 or 10 per cent has been allowed. Other impurities should not amount to more than 2½ per cent, and the $CaCO_3$ content should be at least 95 per cent.

Other requirements for limestone are that it shall be virtually free from mica, pyrite, and graphite or other carbonaceous material, and that when dissolved it will not form a dark scum on the surface of the solution nor form a sludge.

Exact figures on the quantity of high-magnesium lime sold to sulphite paper mills are impossible to obtain as many lime producers who sell burned lime to paper mills do not specify the kind of paper mill, and crude dolomite may be bought by paper mills and calcined at the mill. The following figures are an estimate of the quantity and value of high-magnesium lime sold to sulphite paper mills from 1920 to 1929.

High-magnesium lime sold by porducers in the United States to sulphite paper mills, 1920-1929

Year	Short tons	Value
1920	121,817	$1,347,321
1921	68,992	711,345
1922	70,000	650,000
1923	75,000	750,000
1924	70,000	625,000
1925	53,000	451,000
1926	52,600	482,000
1927	50,000	441,000
1928	46,000	359,000
1929	51,000	398,000

Lime Mortars and Plasters

Lime manufacture consists essentially in the calcination of high-grade limestone, dolomites, or rocks of intermediate composition. High-magnesium limestones and dolomites

3 National Lime Association, Lime - Its Use and Value in the Industrial Chemical Processes: Washington, D. C., 1930, 88 pp.

4 Claudet, H. H., Limestone for Pulp and Paper Industry: Canadian Min. Jour., vol. 49, April 13, 1928, p. 303.

are used extensively for making lime. The limes of northern Ohio, the most productive lime-burning area in the United States, are nearly all manufactured from a practically pure dolomite.

Lime mortars are used mainly as bonding agents for brick and masonry work, and lime finds extensive utilization in plaster and various synthetic building materials. Such mortar essentially is a mixture of well-slaked lime, sand, and water, to which may be added Portland cement.

The reactions of dolomitic lime and calcium lime differ somewhat in the slaking process. A high-calcium lime expands greatly when water is added and much heat is liberated Such mortars, however, are apt to be short or relatively nonplastic. Dolomitic lime slakes much more slowly, generates less heat, and the expansion in volume is less, which makes a correspondingly smaller yield. As a rule the dolomitic mortars are more plastic and therefore have a greater sand-carrying capacity. The choice of mortars is often determined by the experience of an individual mason or contractor. Dolomitic-lime mortars have a higher ultimate strength, but this factor need seldom be considered in the selection of a mortar. The chief use of high-magnesium lime is as a finishing-coat plaster, for which it is very popular because of its high plasticity.

Blast Furnace Flux

Cost is the principal factor in the utilization of limestone as a flux in metallurgical operations. This is particularly true in the pig-iron industry where large quantities of limestone must be used in the reduction of iron ore. Most furnace operators use the cheapest form of limestone available and tho chemical composition, within certain limits, generally is of secondary importance. In smelting iron ores in blast furnaces the impurities are mainly silica and alumina, and in order to remove those as slag, a basic flux must be added. At the fusion zone about 15 per cent of the original iron is still in the form of unreduced oxide, and if insufficient flux is employed, some of this iron is lost in the slag. The flux also takes care of the ash left from the combustion of coke, and aids in removing sulphur and other impurities. Dolomite is preferred in the manufacture of ferrosilicon and ferromanganese, because it fluxes off very little of the silica or manganese.

The magnesia content of the slag has an effect upon the subsequent use of such material. Some operators prefer 7 to 10 per cent of magnesia in slag when it is used for road purposes. The magnesia content, however, is objectionable in any slag which is to be used in the manufacture of cement.

The effect of magnesia on the fluxing action is a question that has received much discussion. In some instances dolomite and high-magnesium limestone appear to increase the viscosity of the slag, whereas in other cases quite opposite results have been obtained.

Miscellaneous

Glass Works.- Limestone is used in the glass industry as a flux, generally as ground limestone or more rarely as an oxide or hydrate. Magnesium in limestone makes glass somewhat more difficult to melt but sometimes is desired in making certain forms of optical glass. Limestone high in magnesium is said to be preferred where certain types of automatic machinery are employed. Glass manufacturers require a limestone of uniform grade because

of the rigid control necessary in handling flux batches. Therefore, either a high-calcium or a dolomitic limestone is preferred because of the greater consistency in chemical composition of these two kinds of material.

Carbon Dioxide and Chemicals.- A few years ago, according to reports of sales received by the United States Bureau of Mines, crude dolomite aggregating about 35,000 ton and valued at more than $60,000 was used annually in the manufacture of carbon dioxide. Figures since 1926 can not be published because only two operators reported production of dolomite for this purpose. Carbon dioxide is used in the manufacture of beverages and in chemical plants such as those producing epsom salts (magnesium sulphate).

Several process have been pattented for producing various magnesium chemicals from dolomite, including magnesium chloride for use in the manufacture of magnesium metal; but except for plants utilizing carbon dioxide locally, other sources of magnesium, either magnesite or natural salt deposits and brines, are generally considered more economical. Projects utilizing dolomite for the manufacture of Sorel or magnesium oxychloride cements have likewise been doomed to failure unless the raw material so used contained a substantial admixture of magnesite. The amount of dolomite used in the manufacture of carbon dioxide is shown in the following table.

Dolomite sold by producers in the United States for the manufacture of carbon dioxide, 1920-1929

Year	Short tons	Value
1920	33,040	$73,671
1921	34,090	124,984
1922	38,280	58,748
1923	38,460	52,494
1924	34,300	57,105
1925	19,740	36,854
1926	16,520	31,424
1927	(a)	(a)
1928	(a)	(a)
1929	(a)	(a)

(a) Bureau of Mines not at liberty to publish figures.

Agriculture.- Limestone finds wide employment as a soil conditioner and fertilizer. It not only corrects acidity and supplies a valuable plant food but may also add to the porosity of the soil, thereby aiding in proper drainage. For these purposes ground limestone or hydrated lime may be used, and in some instances calcium oxide or quicklime is put on the soil. Opinions differ as to the effect of magnesium compounds, but both magnesium and calcium limestones are used successfully.

Paints, Kalsomine, Whitewash, and Varnishes.- Finely pulverized limestone and marble flour are used to a considerable extent as paint fillers. Lime is employed in the preparation of resins used in the manufacture of varnish, and hydrated lime is used for kalsomines and whitewash. The relative amounts of calcium and magnesium carbonate are immaterial for most of these uses.

Vienna Lime.- Vienna lime is made from high-magnesium limestone or dolomite, analyzing about 55 per cent of $CaCO_3$, and 43 per cent of $MgCO_3$ with traces of iron, silica, and alumina. The rock is greayish white, fine-grained, and contains certain characteristic fossils (gastropods) which are said to affect materially its qualities. The stone is carefully calcined by a secret process, and the lime is cleaned, ground, packed in sealed containers, and sold to manufacturers of buffing compounds.

Vienna lime is used for buffing nickel, brass, copper, pearl, celluloid, and other metal and manufactured articles. It is softer than silica, but generally harder than the artificially prepared metallic oxides such as crocus, red, green, and black rouges. Eardley-Wilmot[5] states that:

> The most important use is for nickel and the lime is now recognized as the standard composition for the "coloring" of nickel after plating, as it gives it a deep "under surface" blue peculiar to the metal. Vienna lime cuts faster than crocus and has almost entirely replaced it for all classes of work. Vienna lime should not be used on aluminum as it attacks it chemically.
>
> The action of Vienna lime in buffing is not clearly known, but it is believed that the heat generated on the buff causes a caustic action due mainly to the lime, and the magnesia does the work as it helps to drag and create the necessary friction, but excess of lime would cause it to slip too readily. As soon as the lime becomes hydrated it ceases to function and also attacks the grease compositions.

Ceramics.- Limestone is used to a certain extent in the manufacture of pottery and porcelain ware. It is used in the form of a carbonate as a levigated natural whiting, as a hydrate, or as an oxide. Its function is to aid in fluxing the various materials and for some purposes a relatively high-magnesia content is preferable. In other cases a high-calcium content is desired.

Prepared Whiting.- Magnesium lime is used in preparing a very fine-grained pigment for coating paper. The iron content must be very low, as a pure white color is demanded. This pigment is made by adding sodium carbonate to a suspension of freshly slaked magnesium lime.

Rubber.- Dolomite or high-magnesium limes are used as hardening agents in the manufacture of soft-rubber goods. High-calcium limes have a similar application in making hard-rubber products. Both kinds of lime are also used to some extent as accelerators in the vulcanizing process.

Tanneries.- At the depilatory stage of leather tanning, lime is used extensively. Ordinarily, any magnesium oxide present is objectionable because it tends to give a "harsh" or "rough" leather. In the manufacture of morocco leather, however, magnesium lime is preferred.

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5 Eardley-Wilmot, V. L., Abrasives, Part I: Siliceous Abrasives: Canada Dept. Mines Branch 673, 1927, p. 97.

Fungicides.- Either a magnesium or calcium-lime hydrate may be used in preparing dry-mix sulphur-lime which is used as a fungicide.

Minor Uses.- Other minor uses include the employment of a limited amount of hydrated lime in pig-iron casting, in "pickling" iron and steel, and in wire-drawing. Some is also used in the manufacture of strawboard. Either a magnesium or a calcium lime may be used.

Uses for Which Physical Properties are Important

Building Stone

Requirements of building stone depend principally upon physical rather than chemical properties. Usually any limestone which has the right physical characteristics such as homogeneity of structure, high density, low porosity, and pleasing color will serve as a good building stone, irrespective of the percentage of magnesium carbonate which it contains. It is held, however, by some authorities that a true dolomite is better than a so-called magnesium limestone because of the slight difference in solubility between calcium carbonate and magnesium carbonate. Sulphurous gases in the atmosphere may cause crystals of calcium sulphate and magnesium sulphate to form which are apt to grow and disrupt the stone. Many deposits consisting of alternate layers of calcite and dolomite do not yield rock satisfactory for building material. The governing factor here seems to be again one of homogeneity.

Many commercial marbles contain varying amounts of magnesium carbonate and some are nearly pure dolomite.

Crushed Stone

The enormous amounts of crushed stone used annually for concrete aggregate, road stone, and railroad ballast make this market outlet one of the most important. Here again physical characteristics are paramount and little attention need by paid to the chemical composition except in so far as experience may prove in a given locality or deposit that a different chemical composition indicates a corresponding difference in physical characteristics. Where this is the case it logically follows that close attention must be paid to any variations in chemical composition. With crushed stone, too, because of its relatively low price, other factors such as availability and proximity to markets are usually more important.

Coal-Mine Dusting

To decrease the hazard of dust explosions in coal mines, finely-ground limestone may be liberally applied to the walls, floors, and roofs of the various entries and rooms. For this purpose the chemical content is immaterial, and either a magnesium or calcium limestone may be used.

Asphalt Filler

Asphalt used for paving and roofing consumes a relatively large quantity of finely pulverized limestone. For this market the physical characteristics are important and particular consideration is given to grinding and sizing. Owing to this added cost of

manufacture, the material commands a somewhat higher price and therefore may be transported to a considerable distance for use. Either a magnesium or calcium limestone may be used.

Miscellaneous Uses

Minor uses of limestone, the requirements of which are similar to those of building stone, include stone used for curbing, flagging, paving, rubble, riprap, roofing gravel, stucco, terrazzo, and artificial stone.

Other minor uses are for filter beds, poultry grit, and lithographic stone of which the German variety is a fine-grained magnesium limestone.

Uses of Limestone for Which Magnesium Carbonate is Usually Objectionable

To outline more clearly the field of usefulness of dolomite and magnesium limestone, the following uses for which a high-calcium limestone is essential or preferred are mentioned.

Portland Cement and Natural Cement

Limestone is one of the major raw materials used in the manufacture of Portland and natural cements. Such stone, however, must be low in magnesium content, the usual tolerance being about 5 per cent of MgO.

Water Purification

Much of the water used for domestic and industrial purposes is treated to remove calcium or magnesium carbonate that has been dissolved and held in solution by carbon dioxide absorbed from the atmosphere. Lime added to the water combines chemically with the excess carbon dioxide, forming calcium carbonate, which together with the calcium carbonate previously held in solution is thereby precipitated and subsequently removed by filtration or by decantation. A high-calcium lime generally is preferred for treating water and may be used either as quicklime or as hydrated lime.

Sand-Lime Brick

Sand-lime bricks are made by mixing sand with a small percentage of hydrated lime (usually about 8 per cent) and then molding to form in heavy presses. Subsequently the bricks are allowed to harden under steam pressure. This results in brick consisting essentially of sand which is cemented together by calcium silicate. For this purpose the lime used must be very low in magnesium.

Silica Brick

Silica brick which is used for a refractory in furnace linings differs from sand-lime brick in the curing process. Silica bricks consist of silica bonded with hydrated lime, after which the mixture is molded, carefully dried, and then burned in a special type of kiln. For this purpose the lime should have at least 92 per cent available calcium oxide and not more than 3 per cent available magnesium oxide.

Alkali Works

Limestone is used extensively as a reagent in alkali works, particularly in the manufacture of soda ash and caustic soda. A high-calcium limestone is commonly employed.

Calcium-Carbide Works

In the manufacture of calcium carbide, limestone is calcined and melted in an electric furnace with carbon in the form of coke, charcoal, or coal. A high-calcium limestone must be used because magnesium is objectionable. The usual requirements are that the stone shall have less than 2 per cent of MgO, 0.01 per cent of phosphorous, and 3 per cent of silica.

Sugar Refineries

Lime is used extensively in the refining of sugar. Magnesia is objectionable and only high-calcium limes are employed. Sugar refiners may purchase raw limestone and burn their own lime. Such stone should contain less than 1 per cent of silica and less than 1 per cent of magnesia.

Cyanide Process

In the recovery of gold and silver by the cyanide process quicklime or milk of lime is added to impart what is known as "protective alkalinity." Experiments with dolomite have shown that the latter should not be substituted for this purpose.

Basic Open-Hearth Steel Flux

A limestone flux varying from 6 to 12 per cent of the basic open-hearth furnace charge is added mainly for the removal of phosphorus and sulphur. As magnesium is a poor phosphorous remover, the maximum permissible content of MgO is usually fixed at 5 per cent.

ORIGIN AND OCCURRENCE

The genesis of dolomites is not thoroughly understood. Some are thought to be of marine origin, formed either directly from magnesium and calcium salts in the sea water, or by a subsequent magnesium enrichment of calcium carbonate already deposited from the ocean by organisms.

Some magnesium limestones and dolomites, however, are of other than marine origin. In certain localities, beds of calcium limestone alternate with strata of limestone containing a greater or lesser percentage of magnesium carbonate. Analyses of a single limestone bed, too, may vary over a relatively small area. The formation of such deposits would indicate that the magnesium carbonate is concentrated either by the leaching away of the more easily dissolved calcium carbonate or by a direct deposition of magnesium carbonate from circulating ground waters; in fact, both processes doubtless have effected changes in some instances.

The concensus of opinion seems to be that most dolomites and magnesium limestones were not originally deposited as such, but that dolomitization resulted from changes subsequent to deposition, mainly by the action of magnesium salts in sea water.

Dolomite and magnesium limestones are widely distributed over the world. In some localities deposits attain enormous proportions, as in the Dolomite Alps of Europe. Many occurrences are associated with calcium limestones, but only few with magnesite. Deposits may be evenly bedded or may be greatly deformed and some have become metamorphosed to dolomitic marbles. Examples of this type are found in Vermont, Massachusetts, New York, Georgia, and Tennessee. Because of the wide distribution of dolomite and magnesium limestone in the United States no attempt can be made here to list such occurrences.

METHODS OF PRODUCTION

The mining or quarrying of dolomite does not differ essentially from that of limestone, and the methods employed for preparing it for market are likewise the same as for limestone and need not be described in this paper. In calcining, however, the presence of magnesium carbonate introduces some special problems. The best grade of lime is produced when the temperature is held as closely as possible to the point where the carbon dioxide is given off freely. For pure limestone this reaction begins at about 900°C. and the heat may be safely carried to 1,200°C. Over-burning is caused by too high a temperature and the resulting product may be dense and discolored, and much more difficult or even impossible to slake. Since the decomposition temperature of magnesium carbonate is about 750°C. and that for dolomite about 850°C., it follows that temperatures which may be safe for burning a high-calcium limestone may overburn the magnesium carbonate and, conversely, temperatures sufficient to calcine the magnesium carbonate or dolomite may leave the calcium carbonate in the raw state.

STATISTICS OF PRODUCTION

Complete production figures for dolomite and magnesium limestone are not available because of the difficulty in segregating them from the total production figures for limestone and lime of all kinds. Production figures are available, however, for crude dolomite used as dead-burned dolomite for refractories, basic magnesium carbonate, sulphite process of paper manufacture, and a certain amount of dolomite which is calcined merely for its CO_2 content. The available statistics which have been compiled by A. T. Coons of the United States Bureau of Mines have already been listed under the respective uses.

Production statistics of limestone and lime are to be found in other publications of this bureau.

MARKET PRICES

The ultimate value of dolomite and magnesium limestones depends mainly upon the extent of preparation for market. Because limestone is widely distributed and is very abundant, it commands a relatively low price, but quarrying, crushing, sizing, burning, and hydrating add proportionately to the cost of the finished product. The market value of such material therefore depends mainly upon the expense of manufacture. Another factor influencing price is the location. Owing to natural conditions some localities are able to produce limestone products at a lower unit cost than others. Transportation of the material is also an important factor in the ultimate value. Where limestone products have to be transported any appreciable distance the freight charges in many cases are larger than the original cost of the material. Production statistics yield an approximation of the unit value of dolomite and high-magnesium lime. It will be noted that raw stone sold by producers for dead-burned

dolomite in 1929 was valued at less than $1 per ton. Dead-burned material for that year had an approximate unit price of about $8.75 per ton. The dolomite sold or used by producers for manufacture of basic magnesium carbonate had a unit value in 1929 of about $1.50 per ton.

High-magnesium lime sold for use in sulphite paper mills in 1929 had a unit value of about $7.80. It must be understood that these prices are very general and do not apply o any one particular operation or locality but are the average of all stone reported to the bureau for the uses described.

The following prices of limestone as quoted in Rock Products of May 9, 1931, may be indicative of potential markets for dolomite:

Crushed limestone, 30 cents to $2.25, depending upon the location and grade.
Agricultural limestone, 75 cents to $6, depending upon source, purity, fineness, method of packing, and whether crushed or pulverized.
Pulverized limestone for coal operators, $2.25 to $6, depending upon locality and grade.
Lime products.- Hydrated lime, $6 to $20.
Ground burnt lime, $6 to $17.50.
Lump lime, $4.50 to $20.
Terrazzo and stucco chips, $9.80 to $13.80.
Chicken grits, 90 cent to $10.

The foregoing prices are per short ton f.o.b. shipping point and depend to a considerable extent upon grade, locality, and method of packing.

IMPORTS AND DUTIES

Dolomite is not specifically mentioned in the tariff act of 1930. Imported for use as monumental or building stone it is dutiable at 15 cents per cubic foot if not dressed, hewn, or polished, and at 50 per cent ad valorem if manufactured (par. 234-c). Dead-burned dolomite, if imported, would doubtless fall within the blanket classification for earthy or mineral substances under paragraph 214, dutiable at 30 per cent on its foreign market value.

Import statistics of dolomite and dolomite products are not available, but in general the international trade on these low-priced commodities is relatively small.

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I. C. 6525 September, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING PRACTICE AND COSTS AT THE VIPOND MINE, TIMMINS, ONTARIO, CANADA

BY

Robert E. Dye

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING PRACTICE AND COSTS AT THE VIPOND MINE, TIMMINS, ONTARIO, CANADA[1]

By Robert E. Dye[2]

INTRODUCTION

This paper, describing the mining methods employed and setting forth prevailing costs at the property, is one of a series being prepared by the United States Bureau of Mines dealing with properties in various mining districts of the United States and Canada.

The Vipond mine is a gold property located at Timmins, Ontario, Canada, and operated by the Vipond Consolidated Mines (Ltd.). For the past fiscal year (ended July 31, 1930) ore was mined and milled at the rate of 310 tons per day, and the value of the bullion produced was $896,397.61.

ACKNOWLEDGMENTS

The author acknowledges the assistance rendered by the mine staff in compiling data for this paper. He is indebted also to Charles F. Jackson, principal mining engineer, U. S. Bureau of Mines, for many valuable suggestions.

HISTORY AND LOCATION

The Vipond mine is located in the Porcupine gold area of northern Ontario, 1 mile south of the town of Timmins. The district, which is approximately 500 miles north of Toronto, in the Hudson Bay drainage basin, is served by the Temiskaming and Northern Ontario Railway. The area may also be reached, except in the winter months, by auto over the Ferguson highway.

Early geological reports[3] mention the occurrence of gold in small quantities, but it was not until 1909 that discoveries were made which led to the immediate opening up of the area.

In the following year, 1910, the initial gold production from the camp amounting to $35,539, was recorded. Since that time much progress has been made. For the year ended December 31, 1929, the total production from the district amounted to $19,281,286. The total production from the camp to the end of 1929 has amounted to $247,392,353.

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2. One of the consulting engineers, U S Bureau of Mines, and general superintendent of Vipond Consolidated Mines (Ltd)

3. Burwash, Edward M , Geology of the Nipissing-Algoma Line: Bull 6, Ontario Bureau of Mines 1896 pp 167-184
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GEOLOGY

The geology of the field has been described in detail by A. G. Burrows,[4] now provincial geologist. The region is located in the midst of a huge peneplane over which passed the Pleistocene glaciers. The local topography is not rugged, the highest hills rising less than 200 feet above the surrounding country. The lesser valleys are well marked, though in their lower reaches they are often muskeg and swamp.

Outcrops are fairly numerous. The overburden, which is usually not heavy, is glacial detrital material consisting of bowlders, sand, gravel, and clay.

The rocks are of pre-Cambrian age, the important rocks locally being a complex extrusive series consisting of basic to acid lava flows which have been extensively and intensively folded and distorted, and later intruded by quartz porphyry. These movements have resulted in extensive shearing and widespread schistosity of the rocks. Ore deposits have been localized in the resulting zones of structural weakness, in part along the margins of the porphyry and in part in various lava beds which have proved to be structurally favorable, due largely, if not entirely, to their physical properties.

The Vipond mine is located at the nose and northern leg of a synclinal fold, the axis of which strikes a few degrees north of east, and plunges toward the east at about 55° from the horizontal. The mine, located as it is in an unusually contorted and distorted area, yields orebodies which are erratic as to occurrence and have a wide variation both as to their attitude and form. This feature will become apparent by comparing Figure 1, which shows the orebodies and mine workings at the 400-foot level, with Figure 2, which shows the corresponding features in the same zone at the 600-foot horizon.

NATURE OF THE ORE

The ore consists of quartz veins and irregular masses of mineralized schist impregnated with quartz. A great variety of sizes and shapes of ore masses are encountered and mined. Variations from the narrowest workable widths up to as much as 60 feet have been dealt with. Lengths vary from 10 or 12 feet up to 400 feet. The dips, generally speaking, are more than 60° from the horizontal, though a few flat veins, standing at about 30° from the horizontal, have been mined. The erratic nature of the ore occurrences makes it vital to investigate by raise or drift every exposure which shows any promise.

The ore is quite hard and stands well. The walls of the deposits are generally well defined but in part are "assay walls." The enclosing rocks, except where they are badly sheared or consist of the graphitic material with which much of the ore is associated, are firm and stand well.

The ore contains 10 to 30 per cent quartz and 5 to 8 per cent of sulphides. Of the sulphides, by far the greater proportion consists of iron pyrite with which is associated small amounts of sphalerite, galena, and chalcopyrite. Only a very small proportion of the gold is visible, by far the greater proportion being very finely disseminated throughout the mass and favoring the pyrite, the quartz, and the schist in the order named.

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4 Burrows, A. G., The Porcupine Gold Area: Ontario Department of Mines, vol. 33, part 2, 1924, 105 pp.

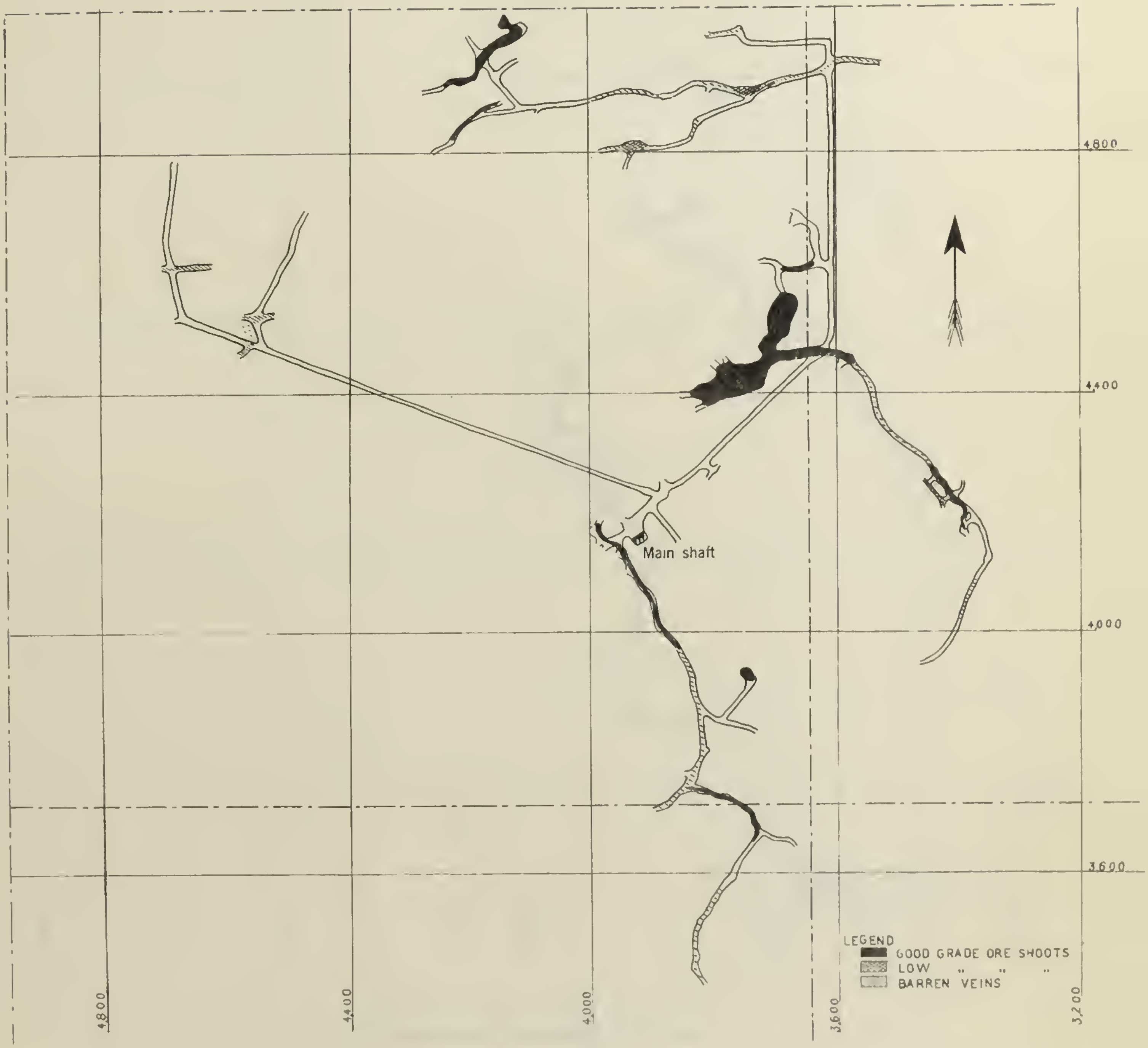

Figure 1.—Orebodies and mine workings, 400-foot level

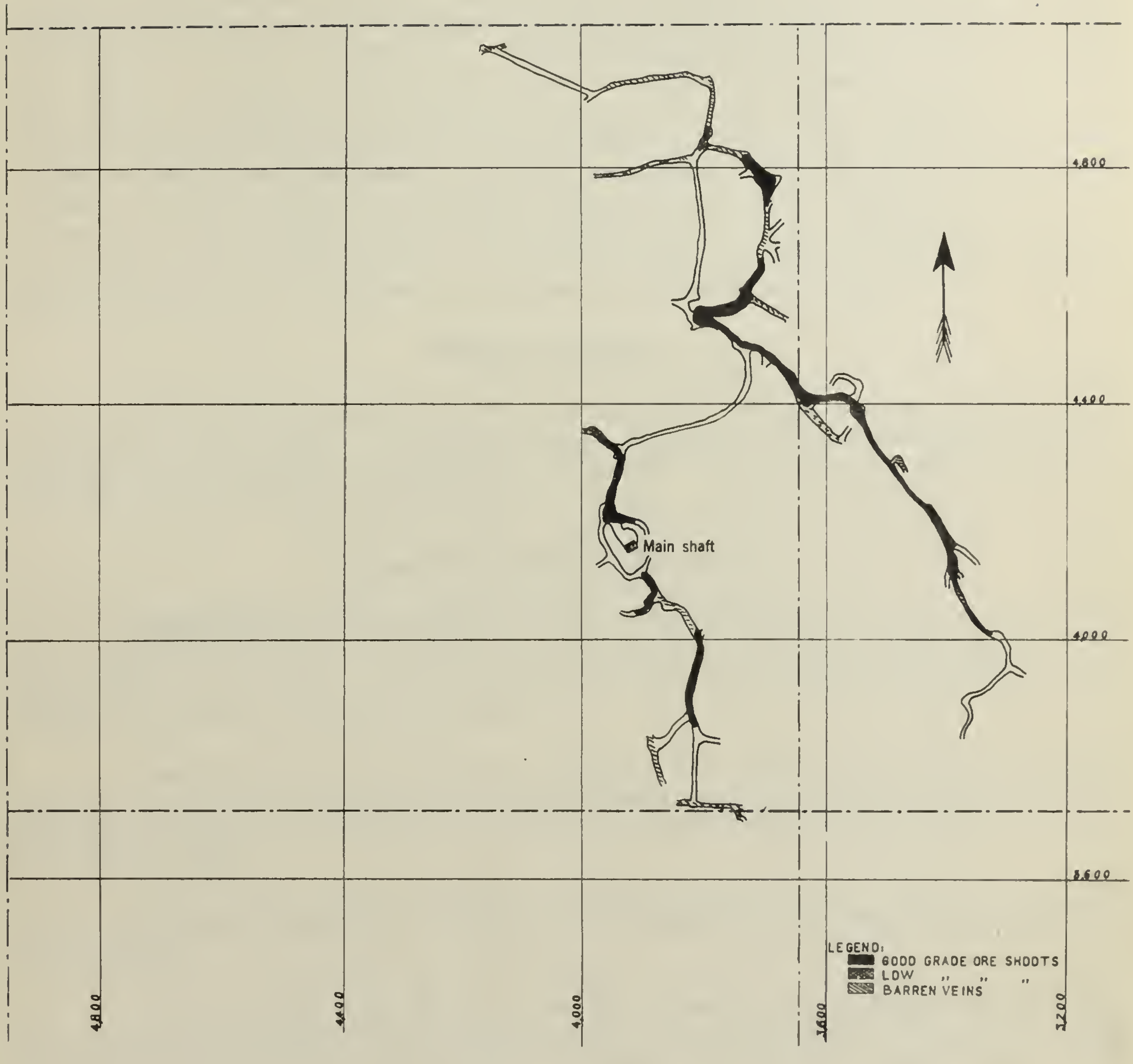

Figure 2.—Orebodies and mine workings, 600-foot level

METHODS OF PROSPECTING AND EXPLORATION

Early prospecting was done by surface trenching, pop shots, shallow pits, an diamond drilling from the surface. Later and more productive exploration has been accomplished by work done from below ground, as by far the greater number of the payable veins apex below the surface. A great deal of diamond drilling, mostly horizontal holes, is done below ground from drifts and crosscuts to serve as a guide for subsequent excavation work. Much valuable information as to structure and possible ore occurrence is obtained from this drilling, though little dependence can be placed upon results as to the probable importance of the ore indications encountered. The drilling serves, at a relatively small cost, to eliminate areas which would otherwise require a heavy expenditure to investigate. The drilling may also lead, and sometimes does, to the development of productive areas which, were it not for this relatively cheap method of exploration, might lie dormant for years. Favorable structures and any ore indications are followed up by excavational work.

METHODS OF SAMPLING

The underground sampling done may be divided into two broad classes, depending upon the purpose of each, namely (1) sampling used as a guide for current operations, and (2) sampling on which estimates of ore reserves are based. The former constitutes the bulk of the sampling and will be discussed first.

Development and Exploration Samples

Diamond Drill.- Sludge samples are taken and assays made for each 10 feet of hole. These samples are usually not of much value, but their cost is small in comparison with the cost of the drilling itself, and the occasional additional information afforded by them warrants the trouble and expense involved.

Diamond-drill cores are carefully examined and logged and are split for assay where at all promising in appearance. A single sample never represents more than 5 feet of core and, if changes in the material traversed are apparent, samples may represent as little as 6 inches of core.

Development Headings.- Crosscuts are rib sampled where any break or ore indication is traversed.

Daily face samples are taken from all drift faces as they progress. Any development rock encountered which will assay $2 or more per ton is sent to the mill, as that figure will cover the milling charge and the tailings loss, and will leave a small profit on the milling alone.

As drifting progresses, the preliminary sampling is followed up by channel sampling of the backs at 5-foot intervals to delimit any commercial ore shoots, as well as to determine any favorable locations for exploratory raises.

Stopes.- As stoping progresses, careful sampling of the walls is done when mining to an "assay wall." When the material is obviously of commercial grade enough breast samples are taken to serve as a rough check against the grade assigned to the ore block based on the back sampling of the drift. A close watch is kept on the walls to locate any branch veins leading off from the main body.

Broken Ore to Mill.- As the ore is loaded at each working place (stope or drift), a grab sample consisting of a handfull of fines is taken from each car and deposited in a box, and at the end of the shift is taken up for assay. While the method of taking these muck samples is very crude and day-to-day samples are subject to considerable variation, the results obtained are of much value in regulating the mill feed and in indicating the grade of ore being currently obtained from various working places.

The value of the ore milled in the past year, as calculated from the muck sample assays, was 4.8 per cent below the actual mill recovery plus the tailings loss.

Estimation of Tonnage and Value

The cross-sectional value of an ore lens at any level is determined by averaging the back samples (channel samples cut from the back and at right angles to the vein) taken after the drift has been slashed out to the full width of the orebody, the samples being weighted according to their spacing and to the width of ore represented by each sample. This average value is calculated after obviously high samples have been averaged with the samples on either side and the resulting value substituted in making the calculation. In arriving at the tonnage and value of the block, the area and value of ore exposed at the level is assumed to extend 50 feet above and below the level, or is joined up with a corresponding exposure at the adjacent level if such an exposure has been developed.

This method of arriving at an estimate of ore reserves without the usual raises from level to level is adopted not because of the regularity of the deposits but rather because of the fact that the deposits are extremely irregular in cross section and in value. while the method often does not result in an accurate estimate of the ore contained in any given block, experience has shown that losses in one quarter are offset by gains in another, so that the method at any given time results in a reasonably accurate picture of the mine as a whole.

Ore reserves are calculated annually, at which time any new blocks are included on the basis outlined above. Partly mined blocks are included for the calculated tonnage remaining unmined and at the value originally assigned to the block, unless the mining accomplished has shown this figure to be obviously incorrect, in which case a new calculation for grade is made. Broken ore is included at a value 10 per cent below the calculated value of the ore in place, unless the "box samples" (grab muck samples) taken in the course of extraction have shown this figure to be obviously incorrect, in which case an adjustment is made, giving due consideration to the value indicated by the muck samples.

METHODS OF DEVELOPMENT AND MINING

General

The property has been developed and is now being worked through a main vertical shaft sunk to a depth of 1,200 feet. Working levels are established at the 100, 200, 300, 400, 500, 600, 733, 866, 1000 and 1200 foot horizons. The shaft is 6 feet by 15 feet 2 inches outside the timbers. It is timbered with 8 by 8 inch square timbers and is divided into three compartments, each being 4 feet 8 inches by 4 feet 2 inches in the clear. Two compartments are used for hoisting (cages in counterblance) and the third compartment is used for a ladderway, pipe lines, and electric cables. The present active mine workings are

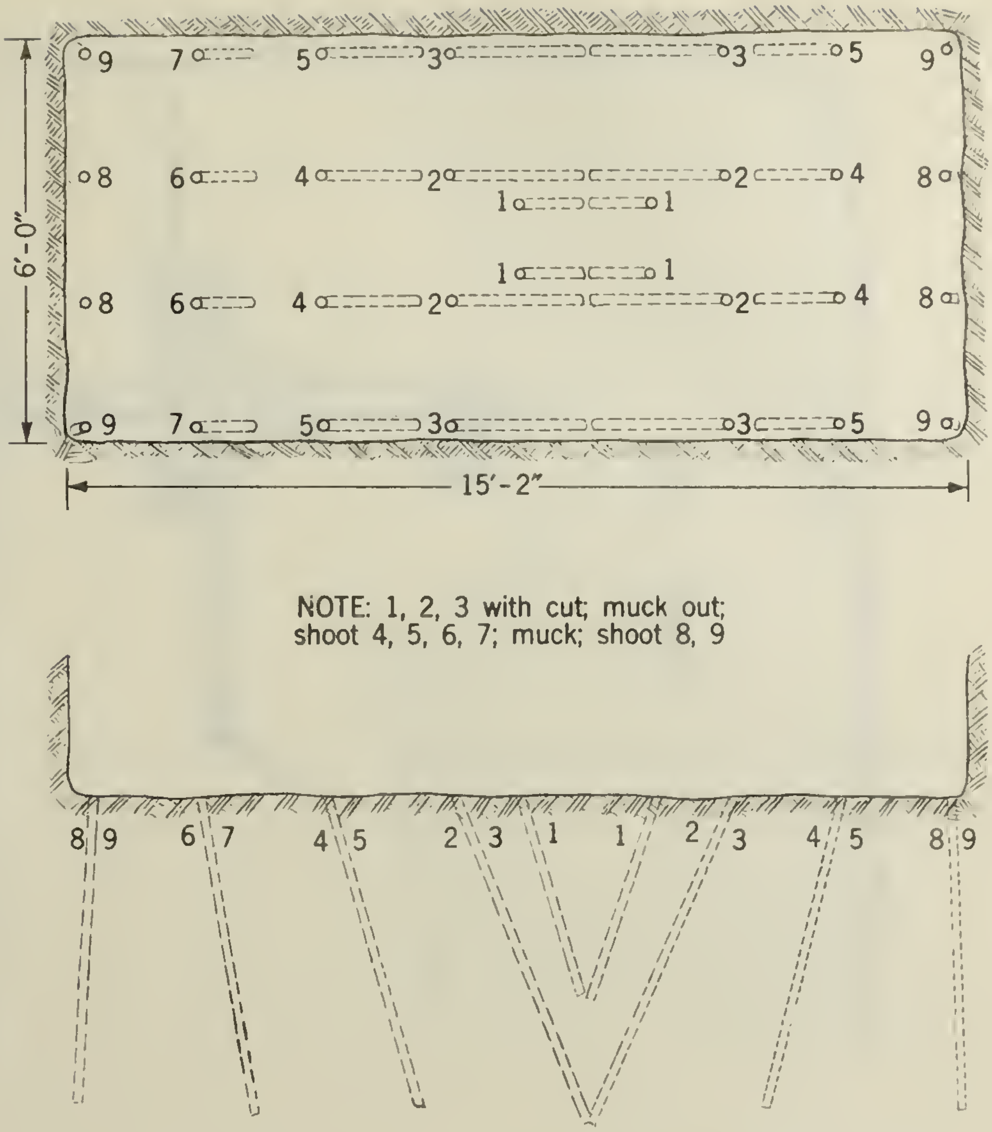

Figure 3.—Standard shaft round

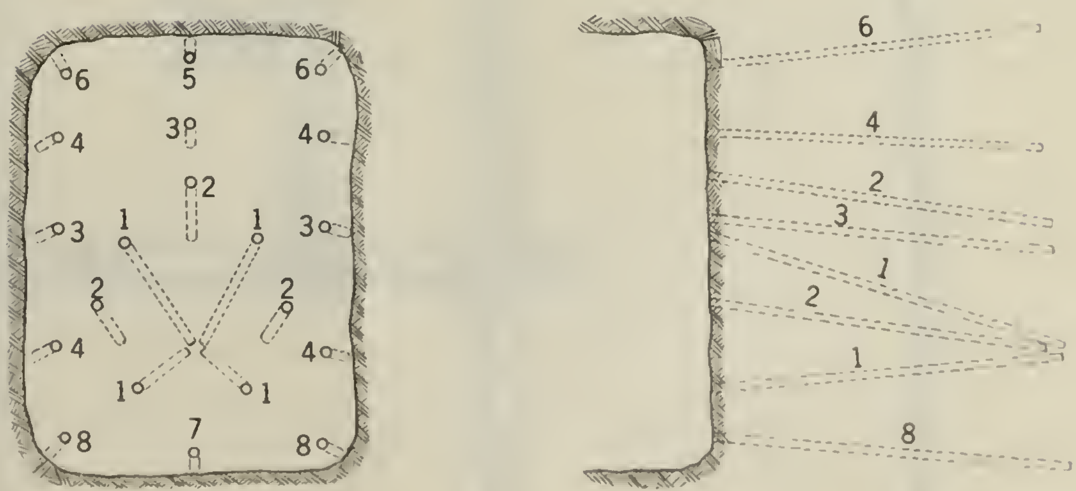

Figure 4.—Standard drift round

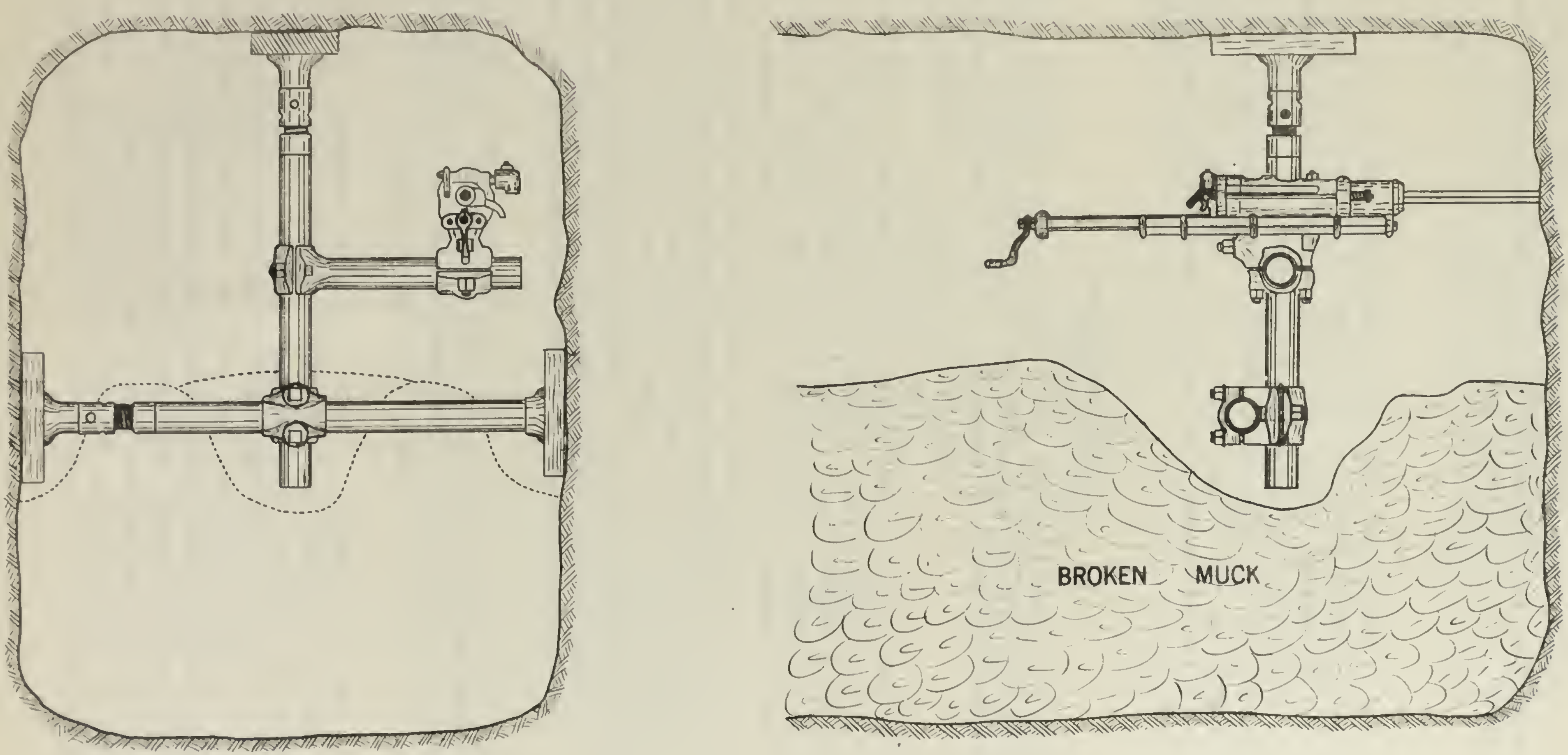

Figure 5.—Crossbar and short column drill mounting

connected by a series of raises and crosscuts to the Old Vipond workings, the shaft of which is 400 feet deep. This shaft serves as an auxiliary exit in case of emergency, as a separate route entirely independent of the main shaft is maintained from the 1200-foot level to the surface. The shaft is of additional benefit at all times in that it greatly improves ventilation.

Current work in the development of the mine is all done from the North Thompson (main) shaft. From each level of this shaft, crosscuts are driven to the breaks or veins in which ore lenses are known to occur, and drifting is done to determine the payable portions which are later extracted. From the drifts and crosscuts resulting from this work, a diligent search is made by diamond drilling and by subsidiary drives for new lenses not previously exposed on other levels. The irregularity of the ore occurrences as shown by Figures 1 and 2, previously referred to, makes it apparent that no uniform lay out for all levels is possible.

Development Details

Shaft.- The shaft is at present being deepened below the 1200-foot level. A standard shaft round is shown in Figure 3, the numbers indicating the order of firing. Owing to limitations in hoisting capacity, it is necessary to do the work in 24-hour cycles, which must be strictly adhered to. A cycle is as follows:

The day shift (7:00 a.m. to 3:00 p.m.) drills off the round and blasts the cut, holes No. 1, 2, 3 and removes the resulting muck. They reblast the cut if necessary and blast holes 4, 5, 6, and 7 when coming off shift. The afternoon shift (3:00 p.m. to 11:00 p.m.) mucks as much as possible and when coming off shift blasts the remaining holes of the square up. The graveyard shift (11:00 p.m. to 7:00 a.m.) clears the shaft of muck, picks the bottom, lets down the machines and hose, and makes the air and water connections ready for the drilling to be done by the day shift. All the drilling is done on the day shift and by the same crew of five men operating drifters with 3½-inch cylinders, equipped with handles and used as pluggers. The crews, composed of five men each on the afternoon and the graveyard shifts, change shifts each week. All the men employed in the shaft are miners and do their own timbering. A 5-foot round is pulled six days per week and the timbering is done on Sunday. Blasting is done by electric time fuse and battery.

Drifts and Crosscuts.- A standard drift and/or crosscut round is illustrated in Figure 4. Cut holes No. 1 are blasted first and if it is found necessary they are reloaded and blasted with the square up. Where possible, a clear heading is furnished for drilling: that is, an advance of one round per 24 hours is made.

In certain development headings, which it is desired to rush, an advance of two rounds per 24 hours is made. Figure 5 shows an arrangement of crossbar and short column which has been found useful in avoiding "mucking back" to clear the heading for drilling, where advancing two rounds per 24 hours. The crossbar is set in place about 3 feet below the roof of the crosscut and above the muck from the previous round. To the crossbar is fastened, by means of a universal clamp, a short column which is jacked to the roof. Drilling is done off the vertical column in the usual way, the machine itself being mounted on a standard arm clamped to the short column. By the time the top half of the round is drilled off, most of the muck has been removed, and the column is turned on the crossbar and jacked to the floor, and the drilling completed. This arrangement, which may seem a bit cumbersome,

is gladly used by the miners in preference to mucking back for the usual set-up of a column, or making two set-ups of a crossbar. It has undoubtedly added to the ease and certainty of regularly removing the two rounds daily from our double-shift headings.

The usual rate of progress is 5 feet per day for single-shift headings and 10 feet per day for double-shift headings. All mucking is done by hand. A crew consists of a runner, a helper, and two muckers. Blasting is done by cap and fuse.

Raises.- For a raise of 6 by 6 feet, a round similar to the standard drift round is used. For a smaller raise, and especially if in quartz, a "burned cut," illustrated in Figure 6, is used. All raises are driven on an angle of 50 to 55° and without any timber. Usually they are driven from the backs of stopes for a distance of 30 to 50 feet to connect with the level above. When driven from level to level on the vein for exploration purposes, they are driven at the same inclination and without timber, except for the chute and manway at the bottom. This arrangement is illustrated in Figure 7.

Drill Steel Used

One-inch quarter-octagon drill steel with a $\frac{1}{4}$-inch round hole is used. The gage of the bit for various lengths is as follows:

	Length, feet	Gage of bit, inches
Starter	3	$1\frac{5}{8}$
Second	5	$1\frac{1}{2}$
Third	7	$1\frac{3}{8}$
Fourth	9	$1\frac{1}{4}$

Figure 8 shows the bit used, which is a single taper cross bit, and also shows the shank used.

It will be noted that the hole in the steel is punched out to a 9/16-inch diameter at the cutting edge of the bit. This feature improves the drilling speed of the bit somewhat, as it tends to the production of coarse cuttings. It also aids materially in maintaining the gage of the bit, as it keeps the center of the hole high, which tends to keep the drill centered in the hole without excessive wear on the outside cuttings edges of the bit.

The drill is shanked in the sharpener to the form shown in Figure 8 by use of an old dolly machined out to the proper shape. The diameter of the surface which takes the blow of the hammer is made 15/16 inch (1/16 inch less than the smallest diametor of the steel), so that a slight upsetting of the shank will not cause the steel to bind in the chuck of the machine.

Mining

Stoping.- Shrinkage stoping is the method of stoping exclusively used at Vipond. As the ore is broken, casual pillars of ore are left where absolutely necessary, and any waste or nonpayable material encountered in the vein is left and serves to help support the walls while the stope is being scaled and the broken ore removed.

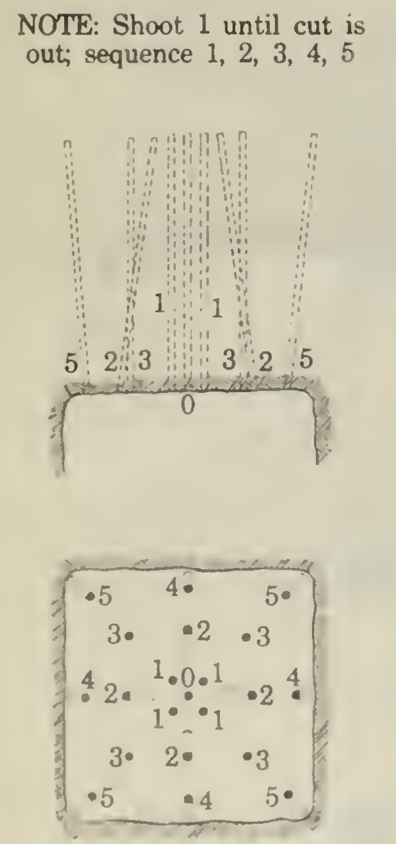

Figure 6.—Burned cut in raise

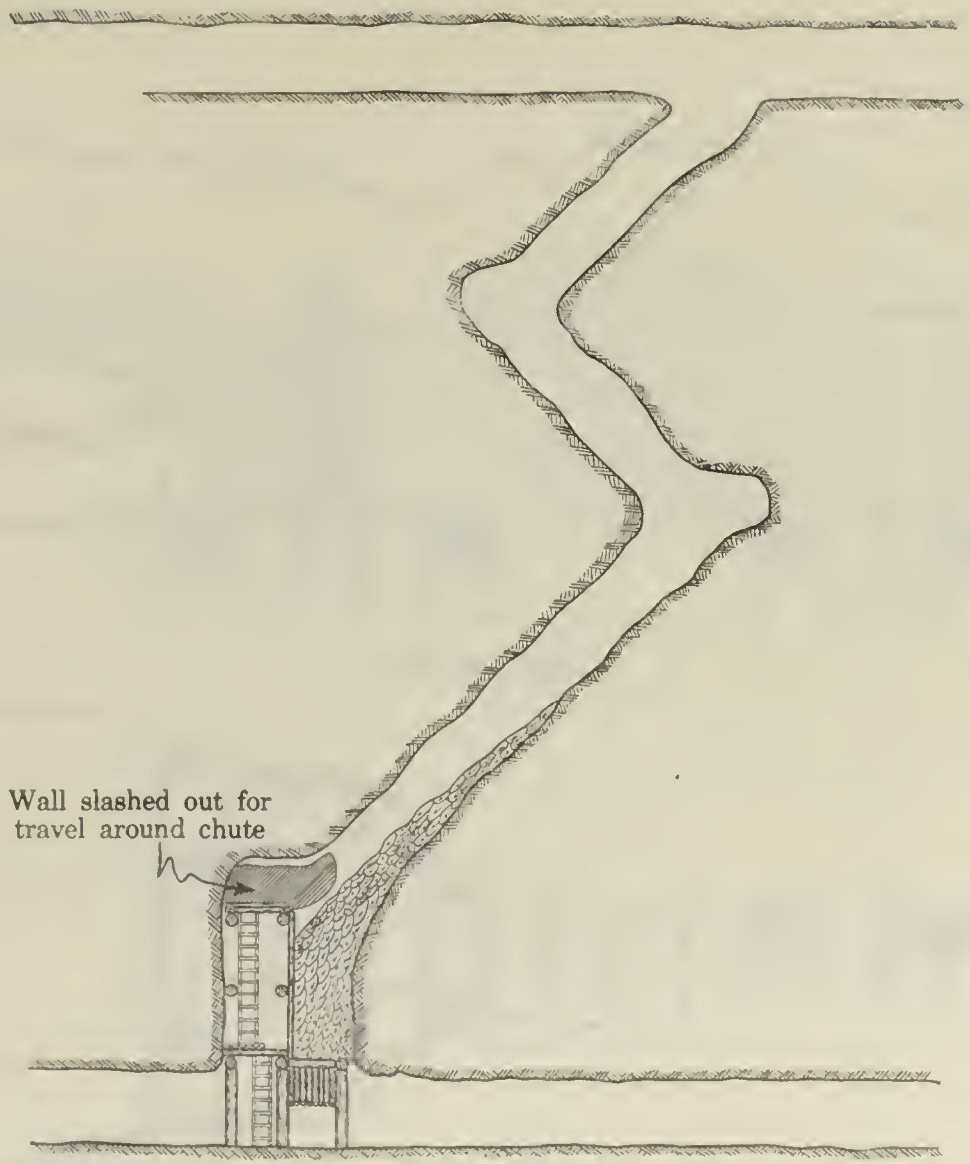

Figure 7.—Exploration raise and chute

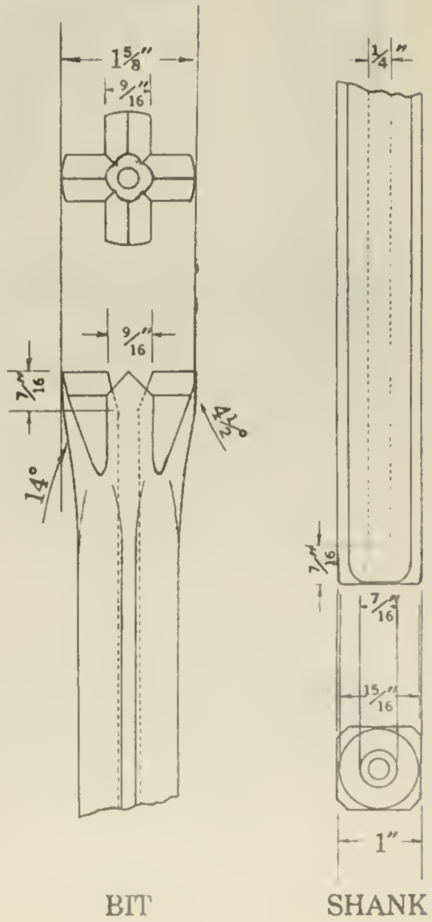

Figure 8.—Drill bit and shank

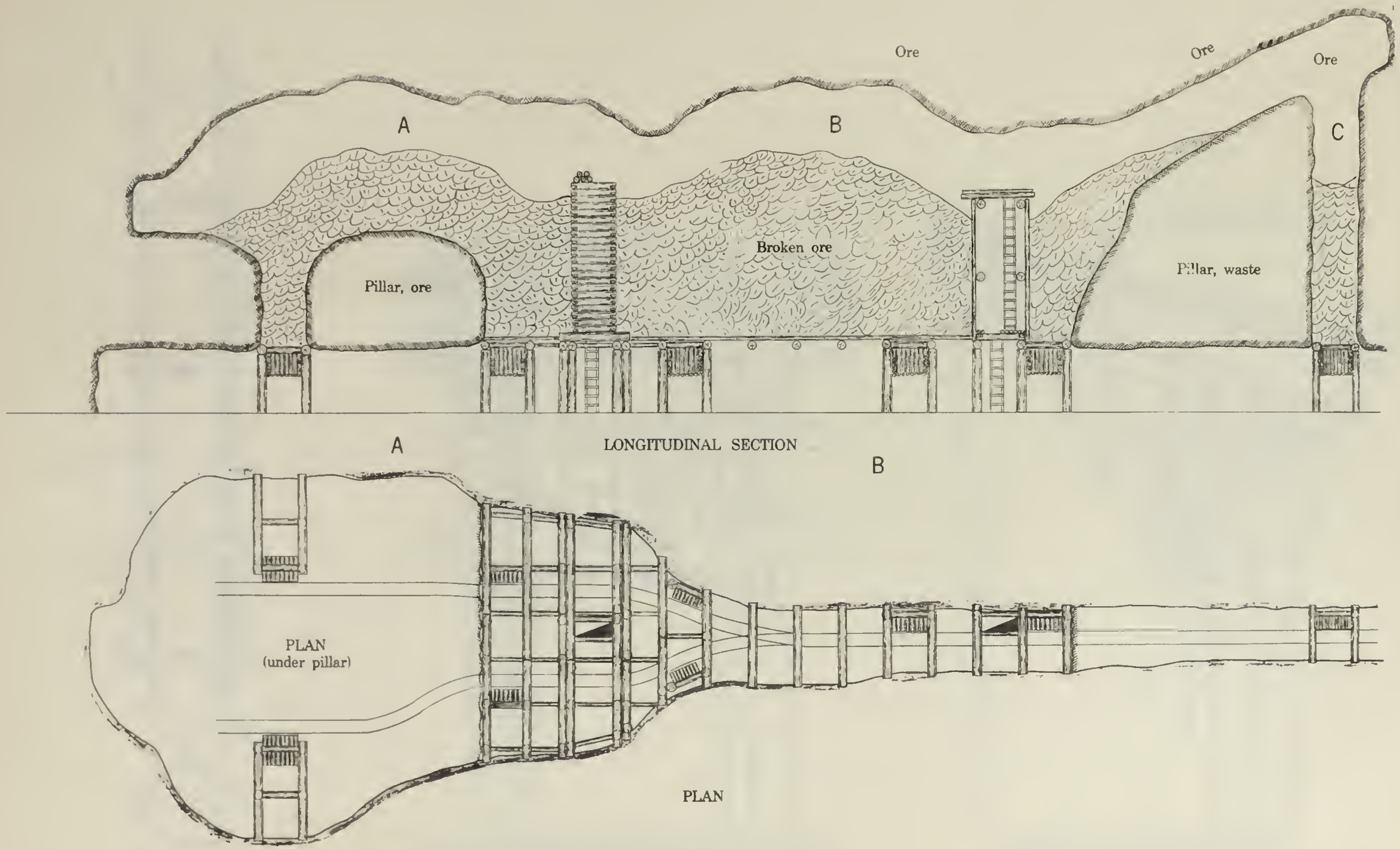

Figure 9.–Stoping in an ore lens of varying width. A, Stoping with box-hole pillars; B, stoping on timber; C, prospect raise up into ore

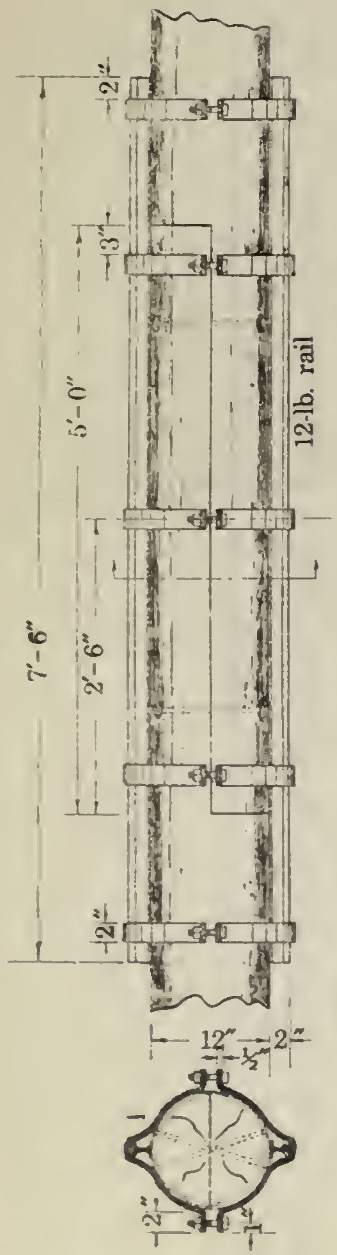

Figure 10.–Method of splicing stull timbers

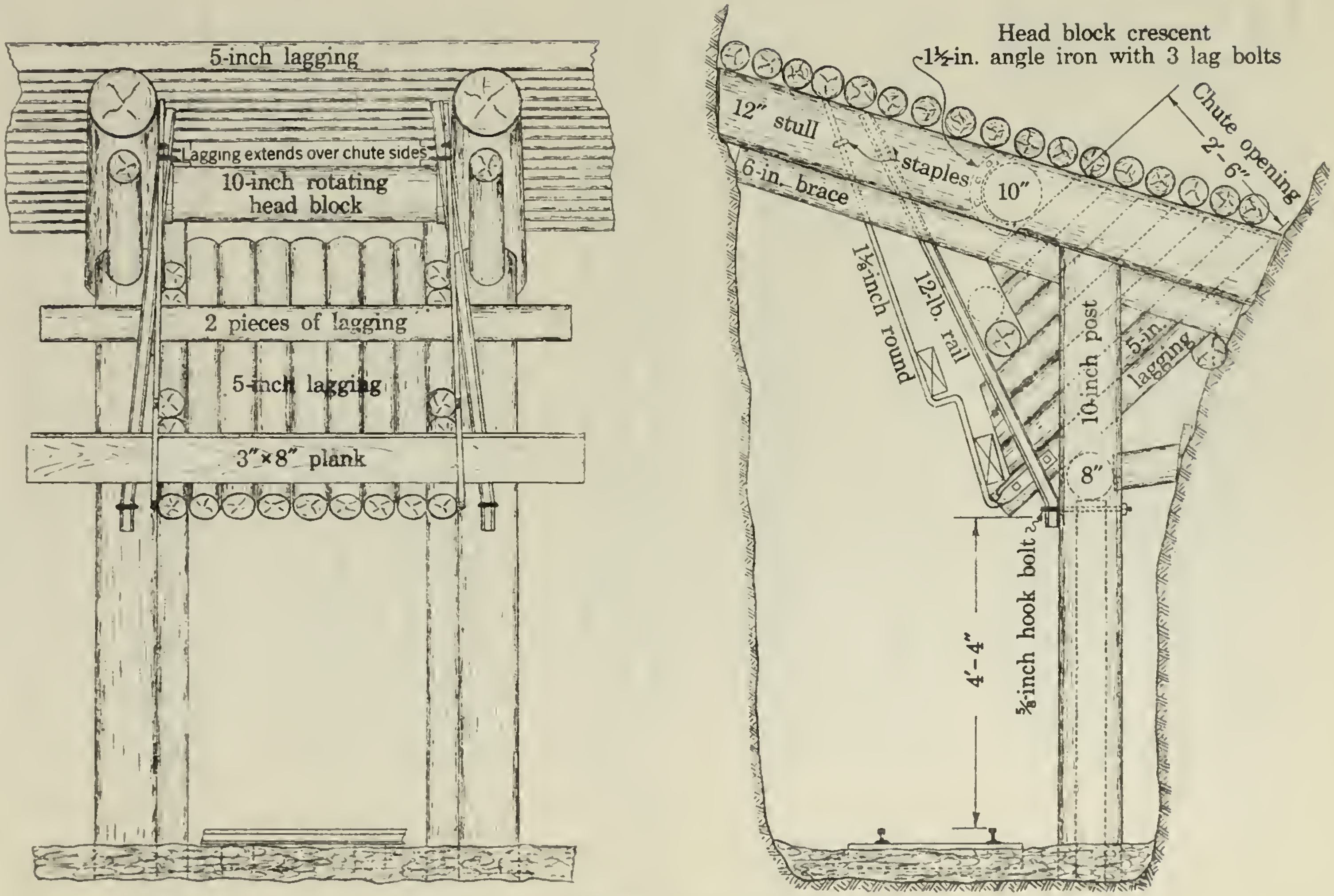

Figure 11.–Standard ore chute

The choice between supporting the broken ore on stulls and lagging or mining it through box-hole raises depends upon the width and grade of the ore. Narrow veins if of payable material are always stulled and lagged, the stulls being supported in hitches. If the vein material exposed at the level is not payable but is of low grade, exploratory raises are driven at favorable points; and if these raises encounter payable ore, a stope is started and the resulting ore drawn off through these raises and any others that it might be necessary to drive for extraction purposes. Ore-bodies up to 25 feet wide are stulled and lagged, the stulls being supported wholly by posts. Ore-bodies in excess of 25 feet in width are opened up by box-hole raises and the roof and floor pillars extracted after the stope is drawn empty.

Figure 9 shows an ore lens of varying width in which each of the methods just described are used.

Where timbering is used, the chutes are placed at 15-foot intervals. Where box holes are put up, they are rarely placed closer than at 25-foot centers, and if they must be driven far to encounter the ore, they may be 40 feet or even more apart.

Each stope is served by cribbed manways at 100-foot intervals. The stope is carried up about 60 feet, after which a raise is put through to the level above and all but one of the manways covered over. The breaking of ore then proceeds through to the level above.

As the broken ore is drawn from the stope, the walls are scaled and bad rock is supported where necessary by sprags or stulls. In narrow stopes only small short sprags are required. In the wider portions more scaling is done to avoid the use of large quantities of heavy timber. Where an occasional exceptionally long stick is required, a splice of two sticks is made as shown in Figure 10 though this expedient is not resorted to if it can be avoided. A stick spliced as shown has very little strength as a beam, but has been successfully used as a sprag, up to 35 feet in length.

The relatively good walls has made it possible to scale all stopes and muck them clear of broken ore. This procedure involves considerable labor but results in the complete removal of the broken ore and in an additional yield from stringers and small masses of ore which were left in the walls as mining progressed.

Mining in stopes is done with Leyner machines drilling horizontal holes. Scalers are equipped with pluggers.

UNDERGROUND TRANSPORTATION

All loading and tramming is done by hand labor. Figure 11 shows the type of chute employed. Ore is transported in flat-bottomed, end-dump cars fitted with Timken bearings. The cars, having a capacity of 1 ton each, are hoisted to the surface on cages operating in balance. The ore is dumped directly into the primary crusher which is located at the shaft head, and the waste is trammed to the waste dump.

PER CENT EXTRACTION

The extraction of the ore is complete except for any pillar or pillars which may be left in the course of mining and which can not later be recovered. Most stopes are finished without leaving a single pillar. The recovery will exceed 98 per cent.

No waste is sorted from the ore either below ground or on surface. The dilution of the ore by waste amounts to about 10 per cent.

COMPARISON OF DIFFERENT METHODS USED IN THE DISTRICT

Until comparatively recently, shrinkage stoping was practically the universal practice in the district. As mining reached greater depths, it was found that the walls did not stand well, and attention was given to other methods of mining. A few stopes were taken out by square-setting but now the cut-and-fill method is becoming the more general practice in those cases where shrinkage stoping is not used.

WAGES PAID, CONTRACT AND BONUS SYSTEMS EMPLOYED

All work below ground except in drifts and crosscuts is done on days pay. The wage rate is: Shift bosses, $7 per shift; miners, pumpmen, scalers, trackmen, cage tenders and powdermen, $4.80 per shift; timbermen, $4.80 and $5.05 per shift; muckers, helpers for machinemen, and helpers for timbermen, $4.24 per shift.

The driving of drifts and crosscuts is done on a bonus system. At one time the work was done on a so-called contract basis by which workmen were guaranteed the regular scale of wages but were given the work to do at a price per foot; they furnished the explosives and the labor for drilling, mucking and tramming. Unless the price paid was frequently changed to meet the varying conditions encountered, it was found that circumstances over which the workman had no control either made it impossible for him to make any bonus at all, or the bonus worked out at a figure which was obviously more than a just reward for any special skill and effort which he might bring to the task. The company found itself in a position where it suffered all the extra cost due to any adverse conditions encountered and did not benefit by unusually favorable conditions, if and when these were encountered. The system now employed does not entirely eliminate this difficulty, but it does to some extent, and it does reward good work under all conditions and at the same time leaves the company in a position to share in the benefits when the work is being done in easy ground.

The bonus now paid is for footage advanced in any month in excess of 4 feet per shift worked, and is as follows:

The drill crew is paid $4 per foot for all footage in excess of an average of 4 feet per shift worked. An allowance of $2.50 per foot advanced is made for explosives and the crew is credited with one-half of any saving they make, or charged with one-half of any explosives used in excess of this amount. The bonus due the drill crew on the above basis is then split three-quarters to the drill runner and one quarter to the helper.

The muckers are paid 50 cents each for footage in excess of an average of 4 feet per shift.

VENTILATION

Ventilation of the mine is almost wholly natural, with the aid, of course, of the compressed air, which is exhausted from the machines while running, and the usual "blowing of smoke" by compressed air between shifts after blasting. In addition to this, a 10-inch fan operated by electric motor has been used on occasion on the lower levels to increase the circulation of air from the shaft through raises or stopes to the upper levels.

The North Thompson or main shaft is downcast, whereas the Vipond shaft, which is at a somewhat higher elevation, is upcast. In the summer months, little or no attention need be given to the rate of air circulation, but in the winter months the natural circulation of air must be checked by means of ventilation doors to avoid trouble with ice in the downcast shaft. It is not unusual for ice to form down to the 500-foot level of the main shaft, and care must be taken to avoid trouble with the cages.

FIRE HAZARDS

The fire hazard in the mine is not considered to be great, though it is recognized that such a hazard does exist in spite of all the care which may be taken to minimize it. The shaft is timbered but is always wet. The stopes require little timber and the ore itself is not inflammable.

To cut down the risk of fire, only one day's supply of explosives is taken below ground each day, all oil is taken below and kept in metal containers, electric wiring is carefully done and frequently inspected, and no inflammable refuse is allowed to accumulate below ground. Cleanliness of workings is insisted upon and made effective as far as possible

Equipment for fighting fire in the headframe, which is of wood construction, consists of fire extinguishers and fire hose connected to the mine hydrants. Little or no provision is made for fighting fire in the mine, though provision has been made to localize the effects of a fire should one occur, and to warn the workmen to leave the mine in case of danger.

At each level provision is made to close the level off from the shaft by means of a fire door. The door is light enough to be closed by one man and hangs from the roof of the drift so that it is out of the way, and there is little danger of the door being abused or damaged so that it can not be used if and when wanted. Various squads of miners are given training in mine rescue work, and the use of standard apparatus at the Provincial Mine Rescue Station located at Timmins.

Provision is made for the introduction of the stench ethyl mercaptan into the compressed air lines to serve as a warning to underground workmen to vacate the mine in case of danger. The United States Bureau of Mines gives an account[5] of the introduction of various stenches into air lines to serve as a danger signal. A consideration of their findings led to tho development of the apparatus shown in Figure 12. These bomb breakers, each charged with a stench bomb, are installed on the main air lines leading to the shaft, and are always ready for instant use. This form of apparatus has the advantages of being cheaply and easily made at any mine, of being absolutely effective as to the introduction of the

5 Katz, S. H., Allison, V. C., and Egy, W. L., Uses of Stenches as a Warning in Mines: Tech. Paper 244, Bureau of Mines, 1920, 31 pp.

stench, and of being so constructed that it may be relied upon to work without frequent inspection.

A test of the apparatus and the stench ethyl mercaptan was made under working conditions at the mine and the results observed by officials of the Inspection Branch of the Ontario Department of Mines. The Ontario Department of Mines Bulletin 71 gives a full account of the results and observations. The test showed the warning signal to be effective.

SAFETY METHODS AND FIRST-AID ORGANIZATION AND TRAINING

In the operation of the mine, the code of rules contained in the Mining Act of Ontario, 1930 (part 8, and secs. 178, 179, and 180 of part 9), are adhered to. No regular inspection is provided other than that done by the mine captain and the shift bosses. Groups of employees are from time to time trained in the practice of first aid, under the auspices of the St. John Ambulance Association. A first-aid kit is kept at the office. As the mine is located near the town, doctors and hospital accomodation are readily and promptly obtainable.

Table 1.- Summary of Costs

Name of mine: Vipond mine. Period: August 1, 1929 to July 31, 1930.

Tons of ore hoisted during period: 113,329. Tons broken: 104,381. Mining method: Shrinkage.

Underground costs per ton of ore hoisted

	Labor	Supervision	Compressed-air drills and, steel	Power cost	Explosives	Timber	Other supplies	Total
Development	$[1]0.362	0.044	0.128	-	0.151	-	0.043	0.728
Mining (stoping)	.626	.084	.250	-	.180	0.046	.089	1.275
Transportation (underground)	.545	-	-	0.046	-	-	.038	.629
General underground expense	.042	-	-	.045	-	-	.061	.148
Surface expense (directly applicable to underground operation)[2]	-	-	-	-	-	-	-	-
Total	1.575	0.128	0.378	0.091	0.331	0.046	0.231	2.780

1 Includes diamond drilling at 8.1 cents per ton.

2 Distributed in various underground accounts.

Table 2.- Summary of Costs in Units of Labor, Power and Supplies

Name of mine: Vipond mine. Period: Aug. 1, 1929 to July 31, 1930.

Tons ore mined and hoisted: 113,329. Tons broken: 104,381.

	Development	Mining (stoping)	Total
A. Labor (man-hours per ton)			
(Breaking (drilling and blasting)	0.212	1.176	1.388
(Timbering			
Shoveling	[1] .212		.212
Haulage and hoisting			.970
Supervision			.107
General			.149
Total labor underground			2.826
Av. tons per man per shift			2.830
Labor, percentage of total cost			56.66 per cent
B. Power and Supplies:			
Explosives (lbs. per ton)	0.73	1.05	1.78
(kind and grade)	(97% - 40 per cent gelatin (3% - 55 per cent ammonia		
Timber (b.m. - lin. ft. or cu. ft.	(.42 bd. ft. (.52 lin. ft. lagging (.18 lin. ft. stulls		
Total Power (kw.h. per ton)			28.0
(1) Air compression	7.1	11.8	18.9
(2) Hoisting	1.1	3.5	4.6
(3) Pumping			3.9
(4) Lighting and Miscellaneous			.6
Other supplies in percentage of total supplies and power			46.0 per cent
Supplies and power, percentage of total cost			43.34 per cent

1 Includes tramming development muck.

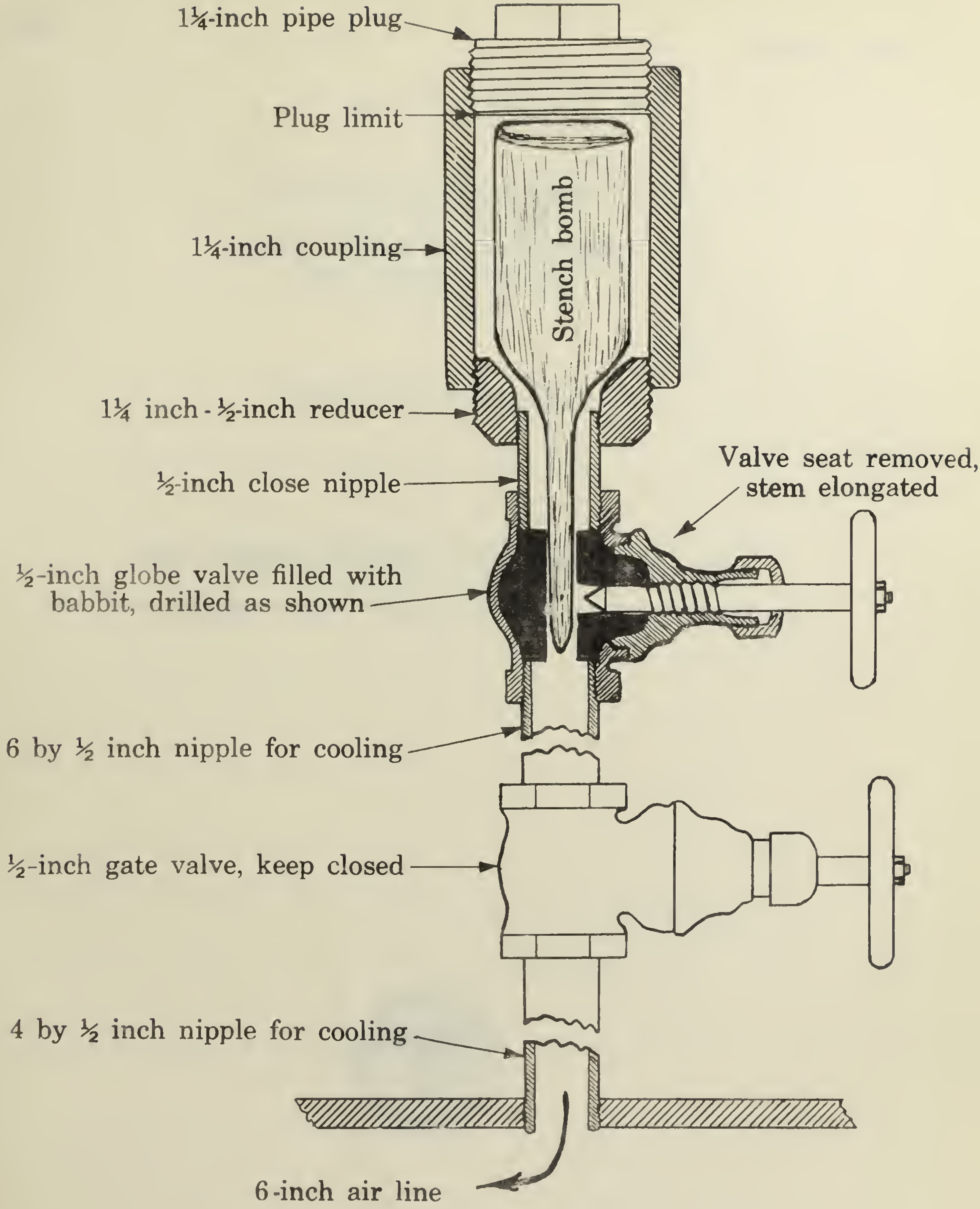

Figure 12.—Stench bomb apparatus

I. C. 6526 October, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

DEEP MINING METHODS, CONGLOMERATE MINE OF THE CALUMET AND HECLA CONSOLIDATED COPPER CO.

BY

Harry Vivian

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

DEEP MINING METHODS, CONGLOMERATE MINE, CALUMET & HECLA CONSOLIDATED COPPER CO.[1]

By Harry Vivian[2]

INTRODUCTION

This paper deals principally with the retreating method of mining as applied in the Calumet & Hecla Consolidated Copper Co.'s deep Conglomerate mine, pointing out some of the difficulties encountered in deep mining and describing the particular methods used at Calumet to carry on a large-scale operation at great depth with a minimum hazard factor. The circular is one of a series dealing with mining methods and costs which is being prepared for and published by the United States Bureau of Mines.

ACKNOWLEDGMENTS

Acknowledgment is due to those officials and operating heads of the Calumet & Hecla Co. whose years of familiarity with the conditions developing at the Conglomerate mine as great depth was attained have enabled them to contribute valuable information to the subject of deep-mining problems.

BRIEF HISTORY OF THE DISTRICT AND EARLY MINE DEVELOPMENT

The copper district of Michigan is in the extreme northern part of the northern peninsula and extends through Keweenaw, Houghton, and Ontonagon counties, the northern portion projecting into Lake Superior at Keweenaw Point. The copper-bearing lodes are all found within a narrow belt from 2 to 4 miles wide and over 100 miles long, the central portion of which consists of a plateau, running in a northeasterly direction at an elevation of 400 to 600 feet above Lake Superior, from which the ground slopes down on either side, gradually toward the west and more abruptly eastward. In a number of places the plateau is cut across by valleys, the most prominent among which is the Portage Lake Gap, which offers a navigable channel for lake steamers into the heart of the copper district. The waters of Lake Superior on three sides of the region, greatly temper the climate.

Michigan was admitted into the Union as a State in 1837. The first information relative to the occurrence of native copper on Lake Superior was furnished in a report by Dr. Douglass Houghton, State geologist, submitted in 1841 to the State legislature.

In 1843, by a succession of treaties with the United States, the Chippewa Indians relinquished their claims to the lands of the upper peninsula of Michigan, and immediately upon the issuance by the War Department of Government permits to individuals to explore for

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6526."

2 - One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, Calumet & Hecla Consolidated Copper Co.

for minerals and locate tracts of land, a wild rush commenced to the district, even before the completion of the linear survey of the public domain, which was then being made in the State of Michigan by the United States Government.

This system of permits was found to be very unsatisfactory and was abandoned in two years. Thereafter land was sold as surveyed and deeded to the purchaser by the Lake Superior Land District Office, established in 1847.

Evidences of copper mining go back to times probably predating that of the Indians, to a race of inteligent people who mined with some system on Keweenaw Point and also on Isle Royale. Masses of copper, some of enormous weight, have been found in old pits which they dug, and in some cases stone or timber platforms were found built in the pits, evidently to assist in raising the masses. Crude implements of stone and copper were usually found in great quantities. In 1844, the first actual mining operations were begun by white men in Keweenaw County, and from that date to the present time, operations have been carried on continuously.

Ninety per cent of the copper of the district has come from six orebodies, one in the Calumet and Hecla conglomerate and the other five in amygdaloids – namely, the Baltic, Isle Royale, Kearsarge, Osceola, and Pewabic amygdaloids. The Isle Royale was opened in 1852, the Quincy and Pewabic lodes in 1856, the Calumet and Hecla conglomerate in 1864, the Kearsarge and Osceola amygdaloids in 1874, and the Baltic lode in 1897. All the ore mineral is native copper, although in the upper workings some native silver was found associated with the copper.

The Calumet conglomerate orebody occurs in a felsitic conglomerate, which is traceable for a great many miles but is found in most places outside of the Calumet & Hecla property as a thin sandstone which in many places is but a few inches thick. At Calumet the bed is from 10 to 12 feet thick near the surface, widening out in the central portion of the property to about 20 feet at the 81st level, which is 8,100 feet below the surface, along the dip. Figure 1 is an isometric sketch of the conglomerate lode. Production from this lode began at Calumet in 1866, and the richest portion was found in the upper areas of the mine. The copper content of the rock remained well above 90 pounds per ton until 1885, 20 years after starting operations. Since then it has gradually decreased to about 40 pounds per ton at present.

From the beginning of operations, the amygdaloid and conglomerate types of lodes have been explored and mined by a great many different mining companies. One hundred and ten companies were actually brought to a producing stage, at a huge cost, which can not be definitely ascertained. Fourteen of these companies returned to their stockholders an amount equal to or greater than the investment, and over 50 per cent of all dividends paid were gleaned from operations in the conglomerate lode of the Calumet & Hecla Co. The total production of copper from the district to the end of 1930 has been 8,403,000,000 pounds.

At one time the copper output from Michigan was over three-fourths of the entire production of the United States, reaching its maximum in 1916 with a production of 269,700,000 pounds. The production in 1930 was 166,400,000 pounds. During the past 28 years, Michigan's percentage of the production of the United States has declined from almost 26 per cent in 1899 to a little less than 9 per cent in 1928.

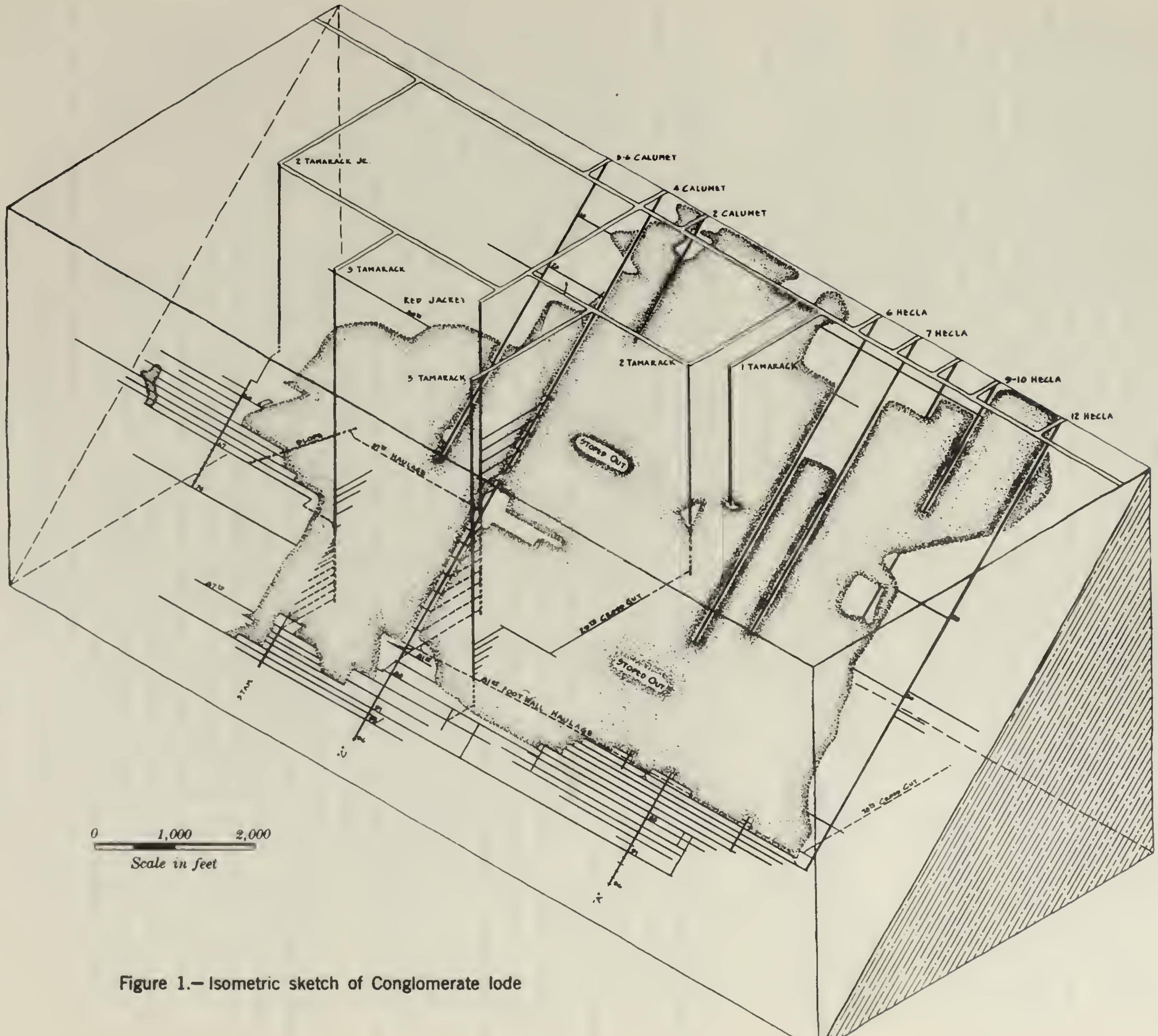


Figure 1.— Isometric sketch of Conglomerate lode

GEOLOGY

The Keweenawan formation of Michigan consists of a series of basic lava flows with interbedded acidic conglomerates and sandstones. The beds strike with the general trend of the shore line of the Keweenaw Peninsula, which juts into Lake Superior in a northeasterly direction, and their outcrops mark the eroded edge of the southern limb of a synclinal formation which dips under Lake Superior and whose northern limb appears again on Isle Royale and on the north shore of Lake Superior in Canada. This series is separated from the Cambrian sandstone on the east by the great Keweenaw fault, which extends nearly the whole length of the peninsula and along which the older series appears to have slipped up and over on the sandstone. The fault has a slightly flatter dip toward the west than the beds of the Keweenaw formation. Throughout the series there are also numerous minor faults where the fault plane cuts across the bedding.

The entire series consists of several hundred separate flows of these basaltic rocks, having a total thickness of several thousands of feet. The copper-bearing lodes are almost entirely in the central portion of the series. Felsitic porphyries are intruded into the series in great mass at several points.

The original source of these flows is considered to have been a welling slit located somewhere near what is now the center of Lake Superior but which at the time of eruption was the crest of a gently sloping plateau. The present dip of the beds, varying from 25 to 70°, is attributed to the tilting brought about by the subsidence of the erupted mass, due to its weight, into the void formed below their expulsion point. As the lavas cooled, quantities of gas collected into bubbles and rose to the surface of the flow. Some of these bubbles were entrapped near the top as the cooling lava became viscous, forming various types of vesicular cappings, as the top crusts of the flows. These top vesicular cappings were the permeable portions of the flows, and the secondary native copper replacing the original minerals in the vesicules formed what is known as the amygdaloidal veins of the district. Most of these flows appear to have followed each other closely, but in some instances the interval between flows was long enough and other conditions were such that felsitic débris from the same original eruptive source was carried along and deposited upon the surface of some of the flows, forming the conglomerate beds of the series.

The favored theory of native copper deposition in both the amygdaloidal and conglomerate beds holds that the great intrusives underlying the series gave off solutions during the period of crystallizing which were rich in copper, arsenic, and sulphur. These solutions being expelled under enormous pressure, followed the permeable channels along the cappings of amygdaloids and through the sandy material, cementing the conglomerate pebbles together. In their upward course cooling took place, and when regions of lower pressure were reached, the solutions reacted with the highly oxidized lode rock; the arsenic and sulphur were oxidized at the expense of the hematite, which is presumed to have been formed by the oxidation of the original iron-bearing minerals of the lodes. The native copper was precipitated, replacing the original minerals which filled the amygdules of amygdaloids or replacing certain minerals forming part of the cementing material between the conglomerate pebbles.

PHYSICAL CHARACTERISTICS OF ORE AND INCLOSING ROCKS

The Calumet conglomerate is a reddish colored bed, varying in thickness from 12 to 20 feet, interstratified with the volcanic rocks of the Keweenawan series and overlain and

underlain by thick beds of trap. The lode consists of pebbles of felsite and quartz porphyry cemented together by small particles of rock, quartz, calcite, and native copper, the rock particles in the cement being similar in composition to the pebbles. The rock is highly abrasive and tough, and has a general hardness of 7. Copper occurs chiefly in the filling between the pebbles or as part of the cementing material, although it is not uncommon to fine pebbles partly replaced by native copper, and frequently fine copper occurs within the pebbles. The bed has a general strike across the Calumet property of N. 33 E., with decidedly little variation. There are so few changes in strike that exploration drifts may frequently be kept in straight alignment for several hundred feet. The lode dips to the northwest at an angle of approximately 38° at the surface, flattening to about 36° at a depth of over a mile vertically.

Within the conglomerate frequently there are found layers of sandstone, usually lenticular in form but quite often of considerable extent. These streaks or beds, greenish gray in color, while commonly barren of copper themselves, usually occupy a position between the commercially mineralized parts of the lode and must be broken with the vein material. The well-marked parting along the bedding of the sandstone requires special attention in placing timber supports.

The entire hanging trap is checkered at various distances apart with slips or jointing faces, nearly all of which are tight and dry, and at angles varying from 30 to 70° with the bedding. Few shatter zones exist, but in places the slips are thickly grouped.

The footwall of the Calumet conglomerate is a highly fragmental amygdaloidal capping on the underlying trap. This noncommercial foot material is in general very friable and contains a large proportion of the softer minerals such as calcite, chlorite, and laumontite in which characteristics it is in contrast to the extremely hard conglomerate vein. When the weight of the hanging wall over any considerable area is permitted to concentrate upon an unmined pillar of vein material, the result is likely to be disastrous due to the thrust of this hard vein pillar upon the weaker foot material below, which causes it to burst upward into the drift opening. This condition was an important deciding influence in the adoption of a retreating system of mining.

The uniformity of the dip and strike, mentioned above, would suggest the absence of many important faults, slips or crossings. However, there are a large number of joints and bedding planes in and along the conglomerate which have a very important bearing upon mining operations, and an extensive system of slips and parting faces in the hanging which cause large blocks to cave for unknown distances above the stopes. A very comprehensive and reliable report, embodying the results of a long-time study of this particular subject, was made by Dr. W. R. Crane of the United States Bureau of Mines and was issued by the bureau as Bulletin 309, in 1929. Reference to this report is suggested for a full and detailed description of the effects of strains and stresses upon rocks, as only a few of the important facts, noticeable in this particular mine, will be included in this paper.

PRESSURE OF BROKEN-ROCK MASSES

The existence of an extensive system of jointing, together with faults, shatter zones, crossings, etc., leave little doubt that all regional strains which were originally inherent in the igneous rock, have long since been relieved, and there is little reason for assuming that any stress other than the weight of superincumbent broken rock masses, acting

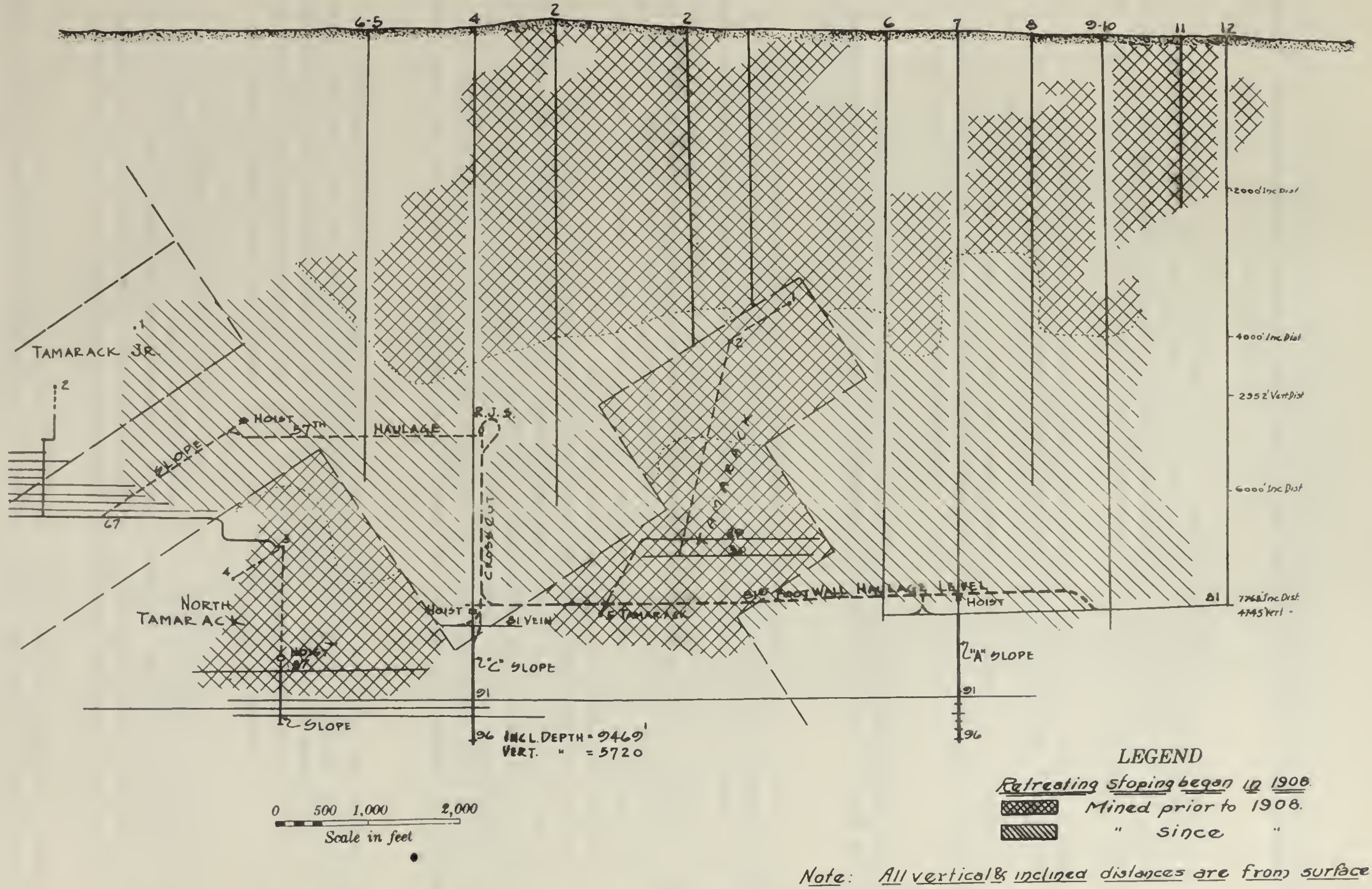


Figure 2.— Horizontal projection of the Conglomerate workings

vertically, is responsible for the tremendous pressure which in the past has caused disastrous rock bursts and pillar failures, and finally, for depths below 3,500 to 4,000 feet has forced the adoption of a particular method of mining.

Major faults and to a lesser degree slips crossing the vein at various angles occupy a very important place in the distribution of the hanging load throughout the active working areas, due to the fact that along a fault plane the rhythmic or uniformly progressing doming action above a caving area may be seriously interferred with. This may result in:

1. A sudden increase of rock pressure upon certain pillars or temporary supports.
2. A concentration of load upon certain points along the fault walls with possible rock bursts resulting.
3. Transfer of a large portion of the support usually given by the thrust of unmined areas bordering the edges of the caved ground, resulting in concentration of a load, greater than normal, upon the active working area.

Bedding planes, joints or slips, particularly those paralleling the dip, when subject to great pressures have been observed to produce a sufficient amount of drag or creep to prevent actual rock burst.

It is not believed that any fixed rule may be applied to ground movement or pressure in the Conglomerate mine, as it is well known that new conditions are being met with daily on any particular retreating face and also that no two nearby stopes present the same conditions at a particular time. A systematic longwall retreating method has been in use in the Calumet & Hecla Conglomerate mine workings since 1908, beginning at a vertical depth of approximately 3,500 feet. This system has immeasurably simplified the problem by reducing the number of variable conditions to be met with and has made it possible to predict in advance an action about to take place. Before the acquisition of the Tamarack Mining Co. by the Calumet & Hecla Co. this property, which consists of an irregularly shaped area extending through about the center of the Calumet & Hecla property from a depth of 3,500 feet to the present bottom, was mined as a separate operation by an advancing room-and-pillar system to a vertical depth of about 5,000 feet. Figure 2 shows a projection of the Conglomerate workings. The variable conditions of pressure, movement, and rock burst were amply demonstrated by this operation, which was discontinued prior to acquisition of the property by Calumet & Hecla.

That the weight of the superincumbent mass of broken rock over the mine workings is directly responsible for air blasts or rock bursts seems to be an accepted fact locally. No idea exists as to the magnitude or even as to the limits of this force except that it is agreed that old crushed-stope areas now open to inspection fully demonstrate that nothing short of a hard, close-grained solid mass will withstand the force permanently.

Depth and the resulting increased pressure are the main causes as proved by the consistent increase in relative severity and frequency of blasts as depth is attained in all fields. While pillar failure is quite common through slabbing at depths from 2,000 to 2,500 feet vertically, actual rock-burst pressures probably do not develop above 3,500 feet on this lode. From 4,000 feet vertically special precaution must always be taken to avoid them.

The two components of the pressure from the overlying rock-mass give immediate evidence of their presence upon completion of a stope. The force normal to the dip of the

vein causes bending and bulging of the hanging, and the force parallel to the dip is evidenced by crushing of the stull headings on the up-vein side where considerable movement takes place by an actual opening developing between the stull and its heading on the down-vein side.

It is believed that immediately above stoped-out areas the overlying rock masses become loosened along preexisting slips or parting faces. The height to which this loosening or breaking up may go depends upon the depth and upon the area mined at that depth. In retreating with approximately 100 per cent extraction of the lode, it is believed that over the extensive areas stoped in this mine, the main pressure block or dome has undoubtedly reached its maximum possible size and weight for excavation of similar thickness. It may be said that it is never possible actually to measure the height to which the overhanging rock breaks up in the immediate vicinity of a deep-stoped area. However, it has been possible in two vertical shafts at Tamarack which penetrate the vein from the hanging-wall wide, and around which the shift pillars were crushed, to gage the height above the vein by the amount of crushing and twisting which took place in the shaft timber sets. The height indicated in these two instances was between 500 and 600 feet, at a time when the shaft pillar, originally 19 feet thick vertically, had been crushed down to approximately 3 feet in vertical thickness by a squeezing out of stope pillars which might safely be said to have been practically completed.

This main pressure block is supported by the solid material at the foot of the arch or dome, both laterally and along the dip; and where a systematic retreat has been in progress over a great area, such as in the case of the Conglomerate mine, with no intervening pillars left standing, the arch may be considered to move downward as the lower support recedes, the lateral thrust following down along the shaft pillars and the upper thrust point gradually following along as the stope filling material crushes into the voids of successively lower openings.

As the pressure block moves downward, various slips and joints are affected, and upon the manner in which the lines of weakness develop depends the amount and kind of temporary support needed in the working area. Undoubtedly this dynamic pressure block at its lower extremity and along its lower sides delivers thrusts of hydrostatic nature. It is doubtful if the hanging load may ever properly be considered static over a retreating stope. The load on the temporary support is directly in proportion to the time the support has been in place for a given position. The failure of a support in one position may be the signal for the immediate failure of other timbers not necessarily adjacent thereto but at points which may be considered to be at the other end of an arched mass of disturbed rock. It may be seen from the above that various combinations of direct pressure, strains and thrusts are at work at all points, and so no exact knowledge may be had as to the nature of the material to be set in motion and no sure prediction may be made as to the distribution of the load.

The heaviest shocks heard and sometimes felt for considerable distances outside the immediate mining areas are rarely located in the mine, and it is probable that they result from the sudden breaking away of large masses at relatively great heights above the hanging wall of the vein during the doming process accompanying hanging subsidence.

METHODS OF DEVELOPMENT AND MINING

The advancing system of mining was used to a depth of 6,000 feet along the lode or about 3,500 feet vertically. While this system of mining was in vogue the practice was to

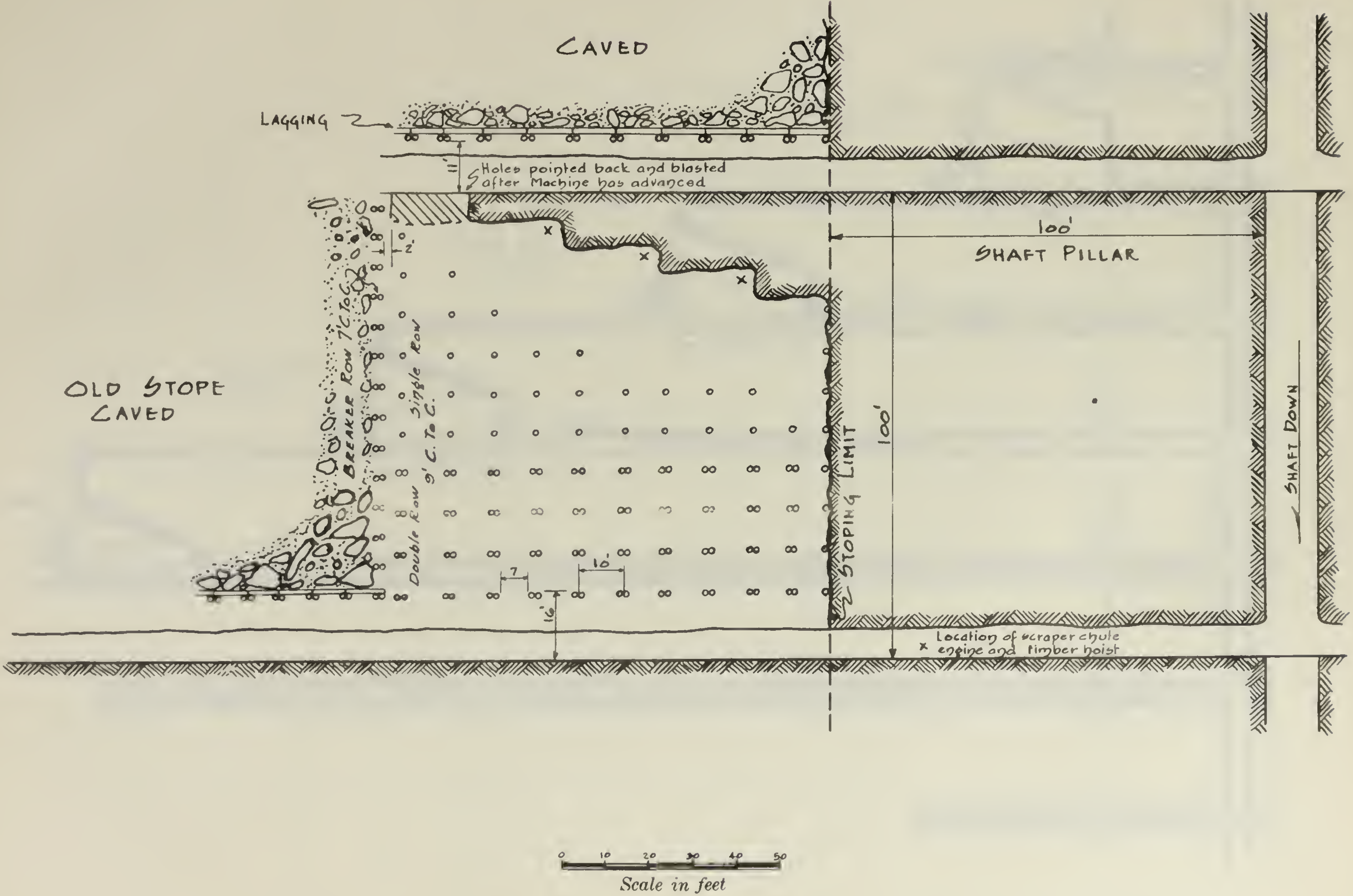


Figure 3.— Open stope, Conglomerate lode

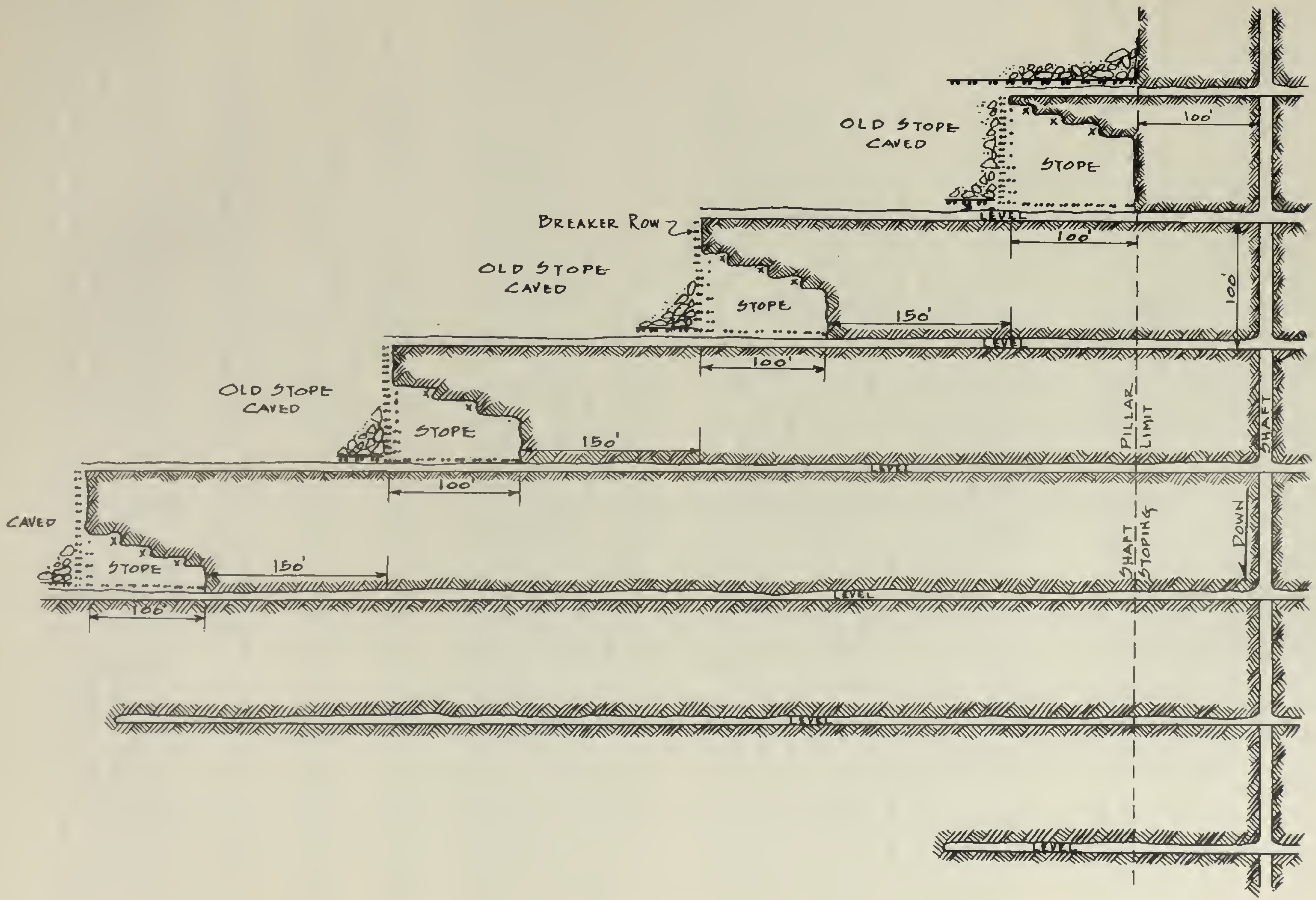


Figure 4.– Retreating stoping system, Conglomerate lode

leave floor arches under each level and above the stope from below. These arches vary in thickness, or rather in depth, along the dip of the lode, from 8 to 15 feet on an average, and constitute lenticular rock pillars, paralleling the strike at every 100 feet in depth. They have long since been crushed by the closing in of mined-out areas above and below, and in 1909 the first attempt was made to mine them systematically. Since that time over one-half of these ancient arches have been retrieved at good profit and with practically no casualties. The closing in of the surrounding stopes has been so complete and the compacting pressure so great that the caved material exposed, as the backs are removed, frequently has the voidless appearance of virgin territory, although in some cases where a sudden breaking away of hanging took place it is necessary to timber to hold the fill while recovering floor arches.

From 6,000 feet to the present bottom stopes at 8,000 feet on the dip (4,900 feet vertically) the retreating system of stoping has been used entirely in all sections of the mine (figs. 3 and 4). This system began contemporaneously with the removal of "old backs or floor arches" in 1909, and from its inception the practice has been to remove 100 per cent of the vein material, mining the floor arches out entirely, as the retreating-stope face progresses toward the shaft pillar. It may be well to say that such a large area has already been mined by the retreating system and the recovery of "backs" has progressed so far above the present active working areas that the effect of any back support from pillars left standing under the old system of mining has long since passed entirely out of the picture. The retreating stope faces may therefore be considered to be, in a strict interpretation, deep workings operating under the maximum pressure made possible by the load of superincumbent rock. The mineralization of the Calumet conglomerate lode has been of such uniform distribution throughout the areas mined that pillars of unminable ore are almost never left standing under present practice.

In mining to greater depths, inclined subshafts have been sunk in the vein, and it is intended to leave pillars of vein material on each side of the shafts to absorb the hanging load (see fig. 1). These pillars have been about 100 feet wide, above the 81st level and are now being recovered, but it is planned to increase this width to 200 feet on each side for the deeper areas below the 81st level. Recognizing the loss due to the deferred recovery of contained mineral values in these pillars, the management gave careful consideration to the problem of establishing the main productive shafts in the footwall before deciding upon the present practice.

The practicability of such a footwall shaft depends upon several factors, important among which are the following:

1. A sufficient assurance of ore continuation to warrant the investment in barren rock work.

2. The material in which the shaft is to be located must be of such character that with a reasonable expenditure for supports it will not collapse.

3. The location of the shaft or of material suitable for the shaft must be such that connection to the vein workings through the intervening barren material will not be so long as to make the cost prohibitive, while at the same time the distance from the vein must be great enough so that during the process of removing the last stope on the vein, immediately above, the ramming effect of this last pillar shall not be sufficient to destroy the shaft or connecting crosscut.

4. The length of time during which the shaft must be kept open.

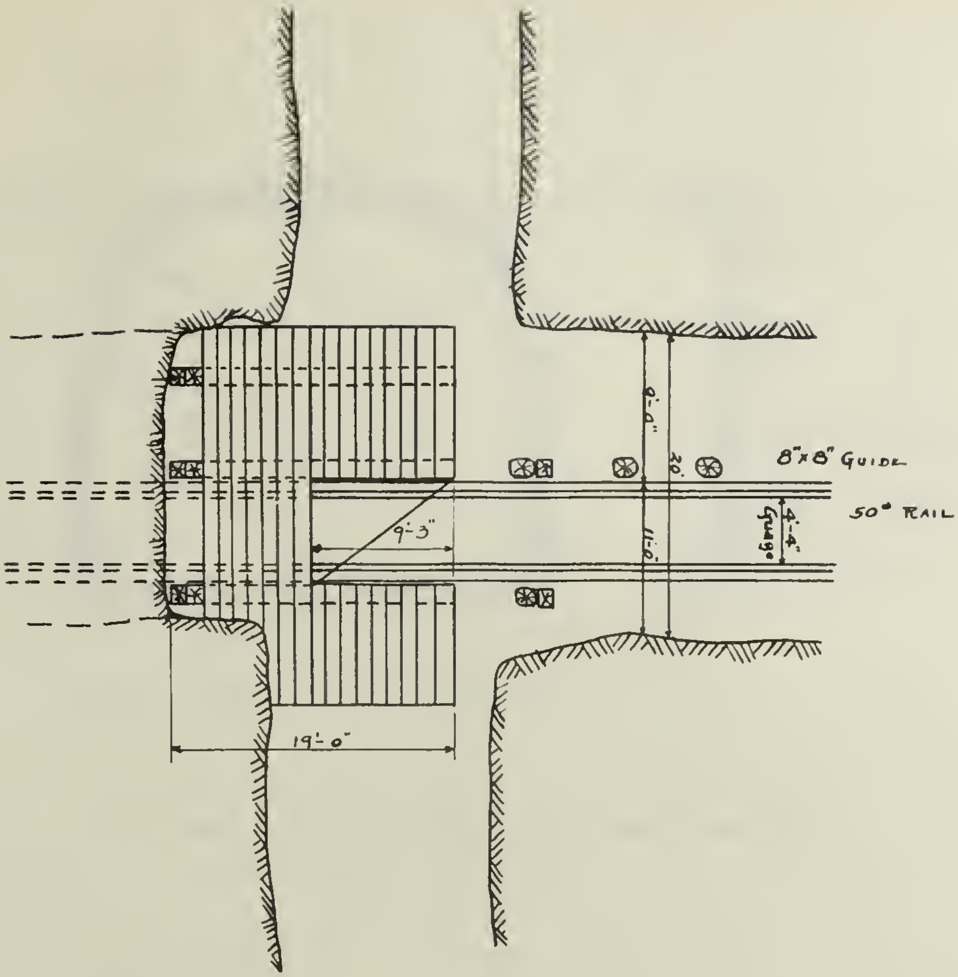
A certain background of experience was available to call upon in the study of deep-shaft location on the Calumet conglomerate lode inasmuch as a block of ground at the north end of the property extending from the 57th to the 67th level had been mined through a shaft situated 90 feet in the footwall. This shaft, approximately 1,200 feet long, while at a considerably shallower depth than the present bottom of the mine, gave constant trouble, due to crushing of the timbers and disalignment of skip ways. Almost constant repairs have been necessary and the entire shaft has been retimbered several times. The original supports were specially designed cast-iron columns and steel I-beam wall plates carrying steel T-rail lagging over the wall plates. This form of construction proved to be very objectionable, principally due to the fact that no cushioning effect was afforded by the columns or dividing timber and that in consequence the material suddenly ruptured without any warning to afford time in which to reinforce the area. Such a condition proved very hazardous and finally the entire shaft had to be retimbered with wood.

In 1919 an experimental footwall shaft located near the "A" incline (fig. 1) was cut in an amygdaloidal lode situated 180 feet in the foot of the conglomerate. This bed was selected in spite of its relatively great distance from the conglomerate because of the homogeneous nature of the material, and its known uniformity of eip and strike, a most important element when it is realized that such a shaft must act as its own pilot during sinking operations. This shaft had been completed for about 300 feet below the 81st haulage drift and connections made to the conglomerate drifts below, when operations were discontinued because of the adverse conditions in the copper industry in 1921. Upon reopening the mine in 1922, the shaft and the immediately surrounding rock were in such a badly shattered condition that any attempt to rehabilitate it was abandoned. As no material of more suitable quality than the bed mentioned is to be found in the foot of the Calumet conglomerate lode, all plans for the future call for shafts in the vein.

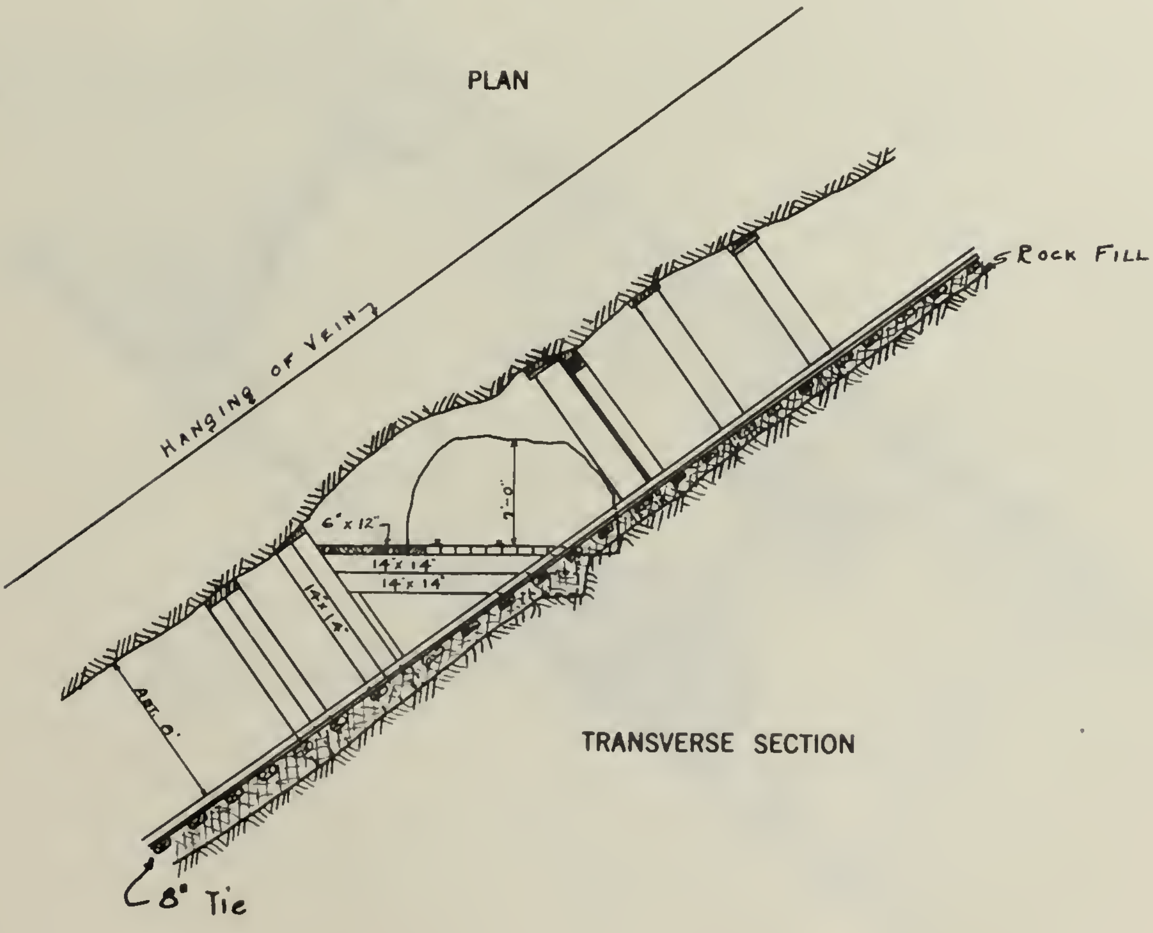
The question has been asked as to the possible advantage in deep mines of sinking to great depths first, next driving development openings to the boundaries and finally applying the system of retreating the stopes toward the shafts, all provided funds were available for such extensive development. Such a plan would undoubtedly reduce the timber cost in stopes and the cost of shaft maintenance, but it is the opinion of the writer that the objections to such procedure far outweigh the advantages, particularly in thin vein deposits of material similar to the conglomerate. Some of the objections may be stated as follows:

1. It is not possible to explore a comparatively thin dipping bed for any great distance below the workings by drilling operations, due to the impossibility of keeping the drill within the vein which ordinarily carries the commercial mineralization.

2. The expenditure necessary to an extended development program together with compound interest upon the same up to the date of realization by stoping operations, is something the average mine operator as owner hesitates to face, even though able financially, due to the undeterminable value of the results to be attained and the well-known hazards of mining enterprises. To make absolute outlay against doubtful returns is the picture which usually appears, and the result is that most deep mine managers prefer to be paid as they go.



PLAN



TRANSVERSE SECTION

Figure 5.— Plan and section of typical plat

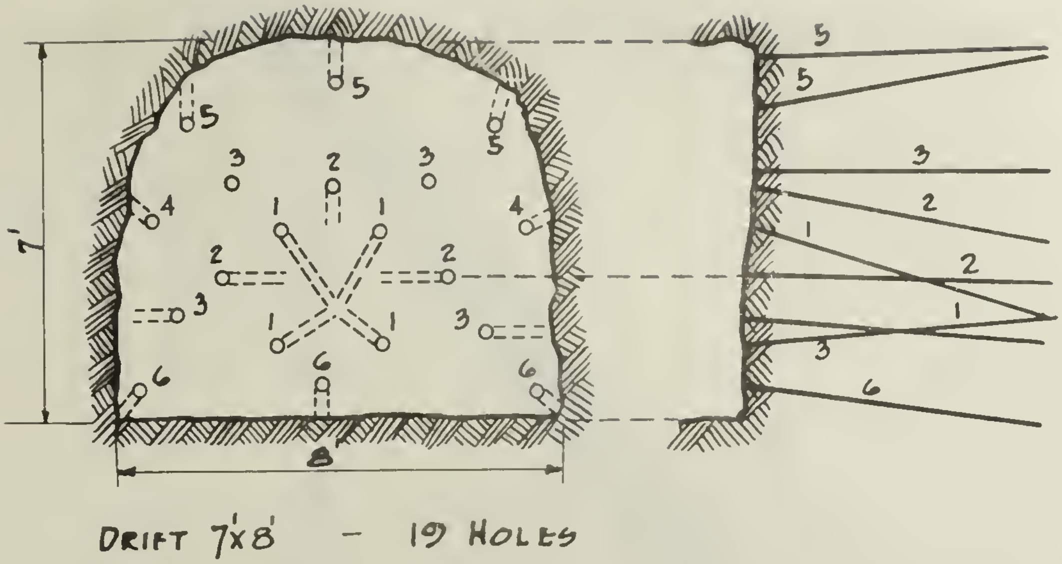


Figure 6.— Pyramid cut used on Conglomerate drifting

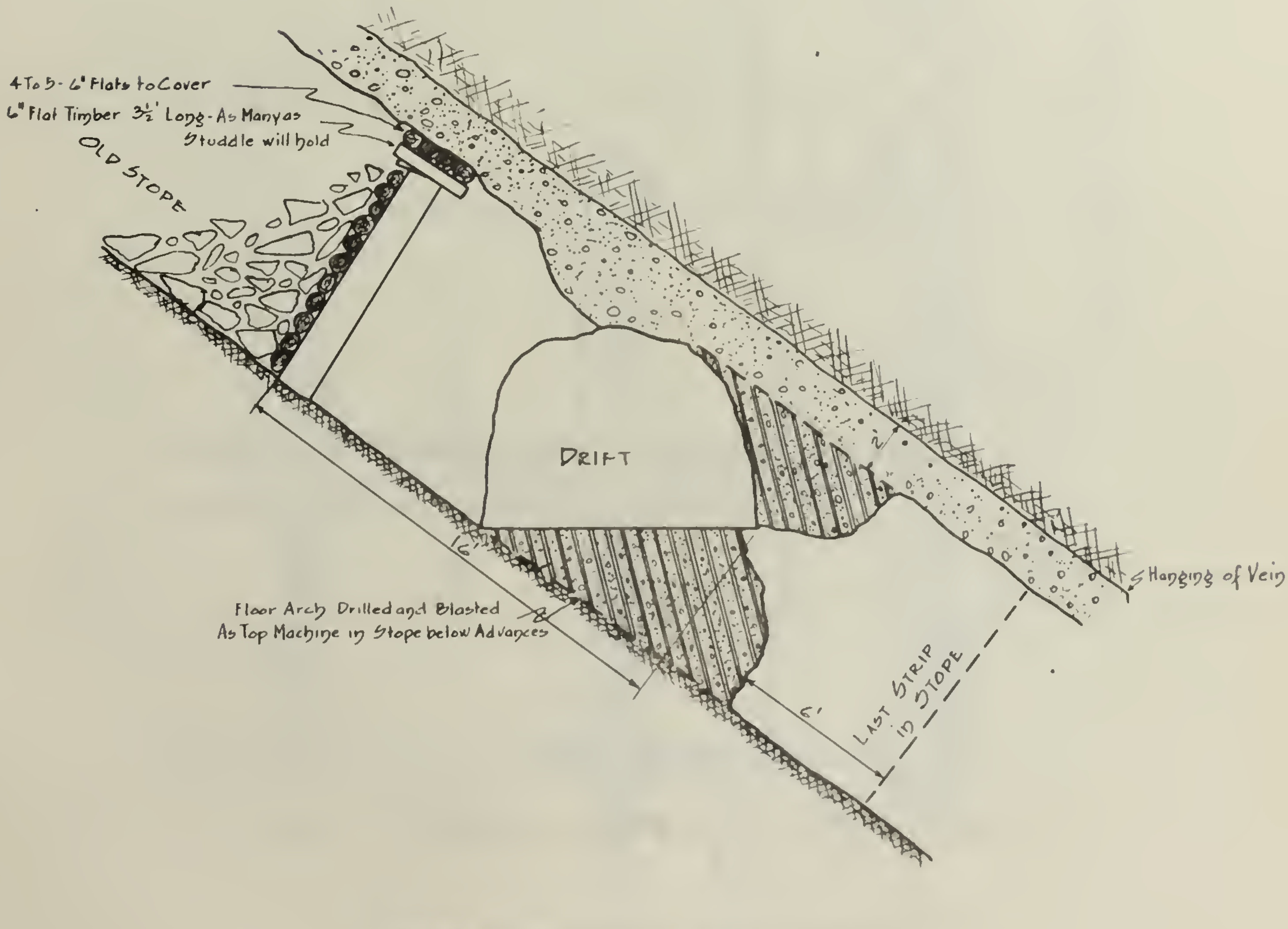


Figure 7.— Section showing removal of floor arch

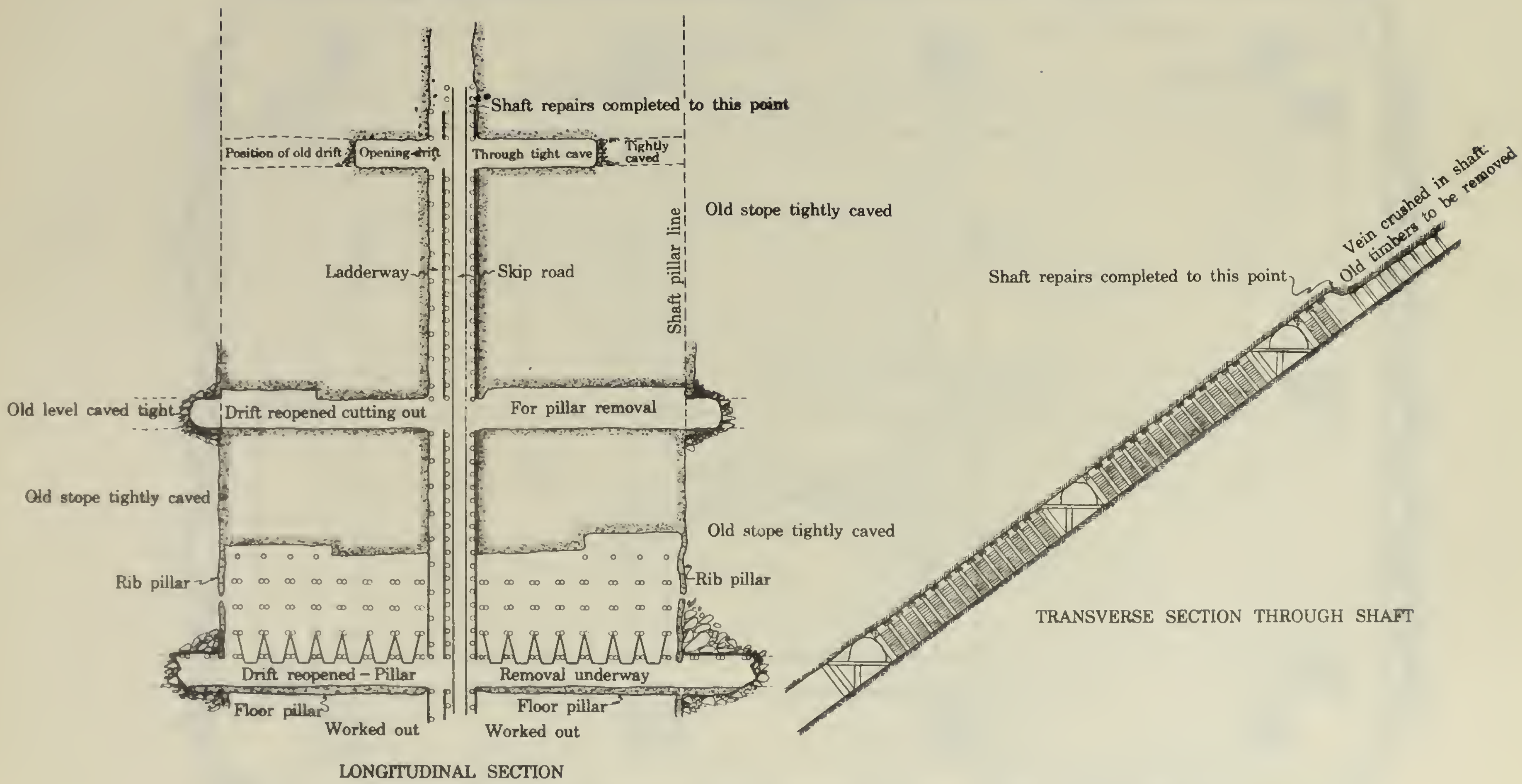


Figure 8.–Method used in removal of shaft pillars

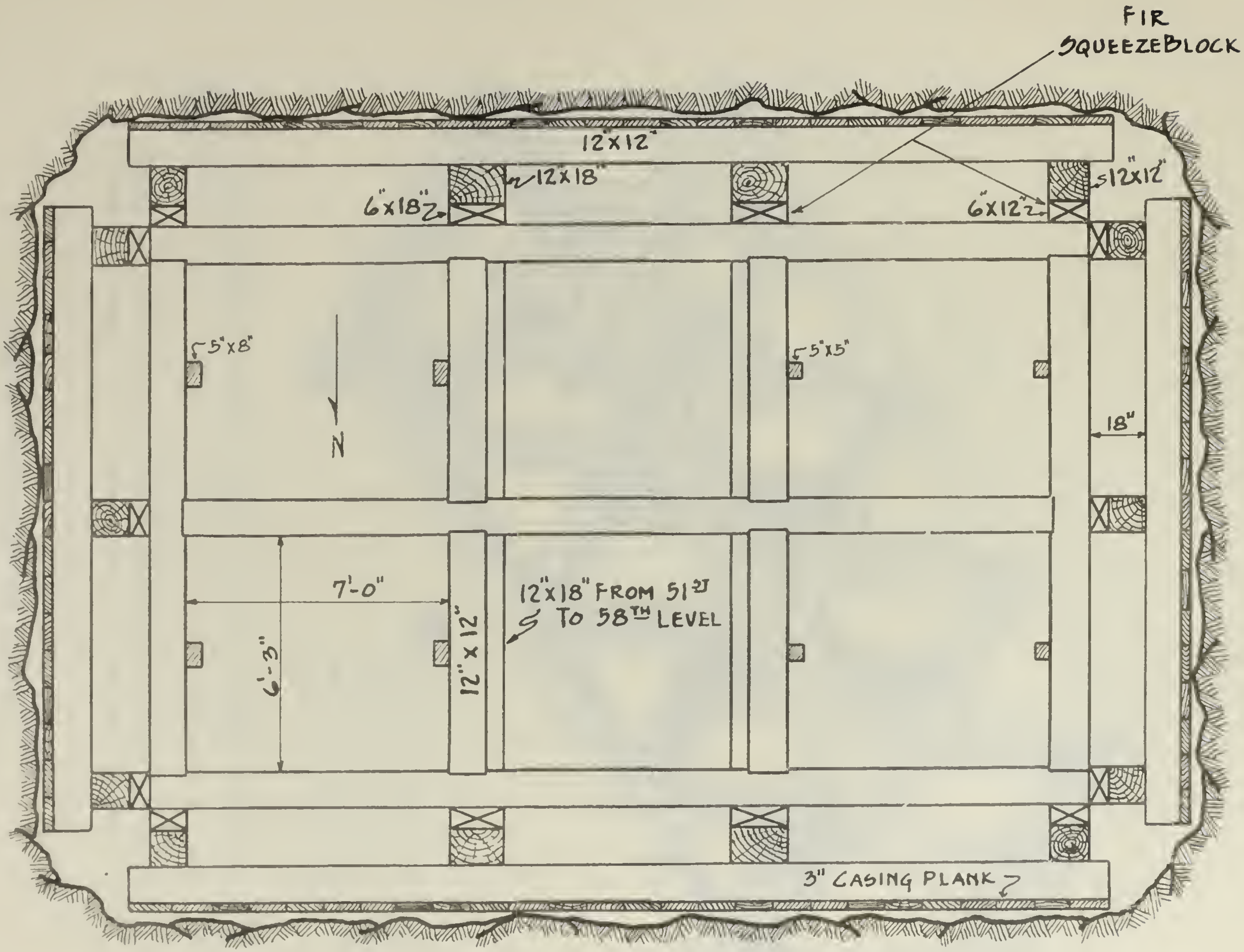


Figure 9.— Umbrella timbering in Red Jacket shaft

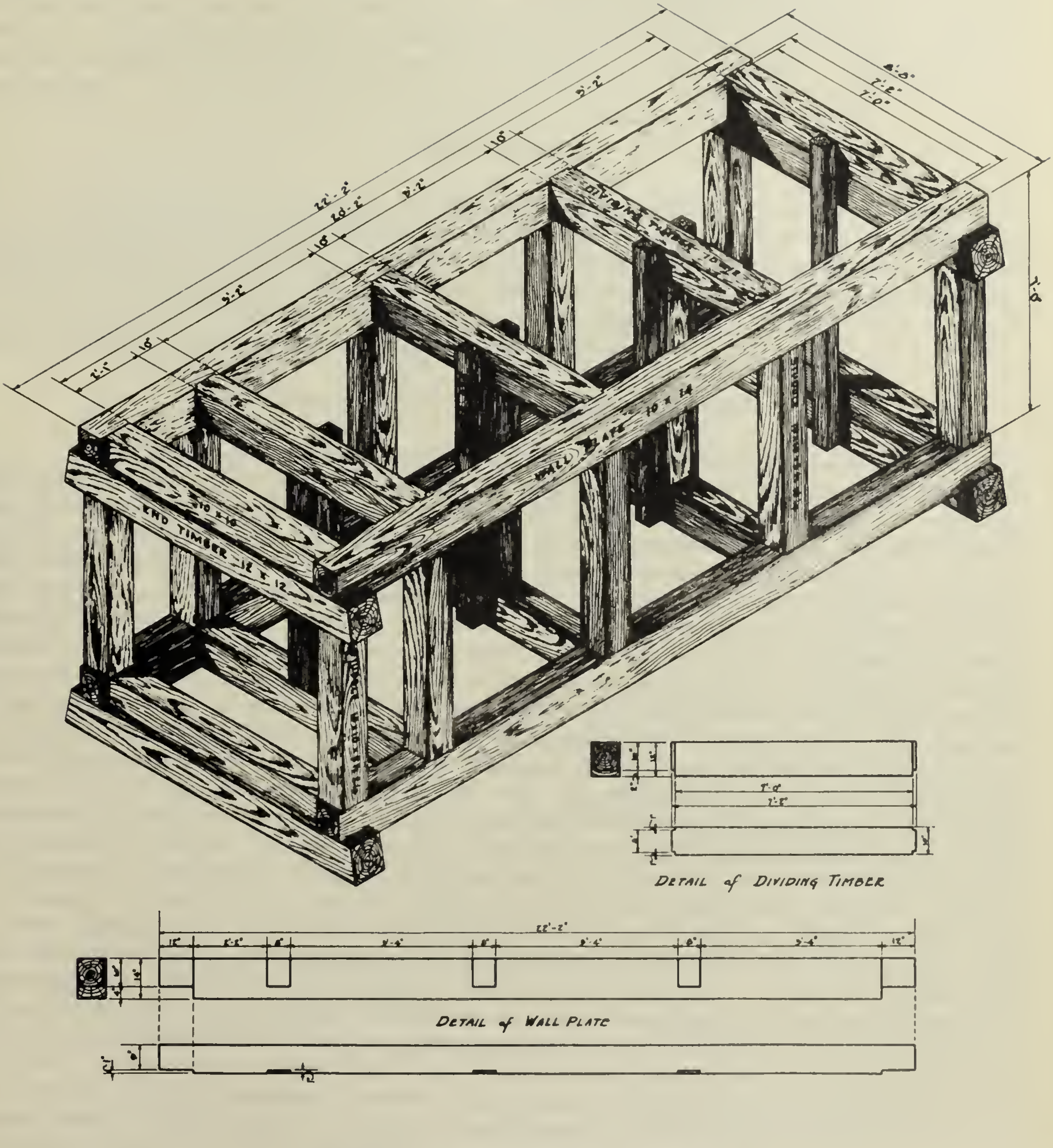


Figure 10.— Shaft timber, Tamarack No. 3 and No. 4

3. All rock in the Conglomerate mine vein must be drilled and blasted. The overlying rock pressure may be said to offer no advantage of any kind to assist in mining the ore. That the reverse is the case is witnessed by the large number of timber supports necessary to make working places safe and also by the frequent and always possible bursts of rock from exposed faces. Serious consideration has been given to mining blocks 400 feet in depth, leaving a 400-foot block above to be mined later from the top down, after hanging pressure has been relieved by the collapse of the top stopes above, and proceeding to great depth in this manner.

SHAFT PILLARS

No rock pillars are ever left standing in the Conglomerate mine stopes. The length of the pillars of vein left standing to protect the operating shafts has been increased continually as depth was attained. In some of the early work at very shallow depths the pillars were about 15 to 20 feet long, gradually increasing to 75 feet for all active openings above the 50th level and to 100 feet from the 59th to the 81st level. It is planned to leave nothing less than a 200-foot pillar below the 81st level.

DETAILS OF SHAFTS AND 81ST LEVEL HAULAGE WAY

By reference to the isometric sketch (fig. 1) and the horizontal projection of the mine workings (fig. 2), it will be seen that the vein has been opened up by a large number of inclined shafts and several vertical shafts, which were sunk to and through the vein and connected to it by a series of crosscuts. The main production shaft is the Red Jacket, 4,900 feet deep vertically and having six compartments timbered throughout with 12 by 12 inch timber sets placed on 5-foot centers (fig. 9). The six compartments are each 6 feet 3 inches by 7 feet, set in two rows of three each, and in addition there is a narrow pump compartment. This shaft reaches the vein just above the 57th level, and at this point, as well as for about 400 feet above, the squeezing effect of the hanging wall load has been severe at times in the past, in spite of the fact that a pillar of vein material 500 feet square was left unmined around the shaft to prevent shaft disalignment. On account of this hanging thrust, an auxiliary set of timber has been employed in this area outside the regular sets. This set, known locally as the "umbrella set" (fig. 9), takes the squeeze and offers a ready opportunity for making shaft repairs without interruption to hoisting operations. Red Jacket shaft connects with the vein by crosscuts on every third level from the 36th down to the 81st. The No. 5 Tamarack, a vertical shaft and next in size, has four 5 foot $2\frac{1}{2}$ inch by 7 foot compartments set in line, and in addition a 3 by 7 foot compartment for pipe. Twelve by twelve inch square-set timbering on 7-foot centers is used throughout. This shaft is used as the main pump shaft for the deep areas of the entire mine. Two large pump rooms, each containing three vertical triplex plunger pumps, operated by 350-hp. motors, are located just off the shaft, one near the bottom, 5,156 feet vertically below surface, the other at approximately the mid-depth point 2,424 feet deep. The bottom room connects with the vein on the 87th level (vertical depth 5,200 feet), the crosscut and part of the drift being used for water storage, which is protected permanently by leaving a block of unmined ore of sufficient size to resist pressure cracks. The upper pumps operate in series with the lower ones, and only nominal sump capacity is required.

No. 3 Tamarack, a vertical shaft, has three 5 foot $2\frac{1}{2}$ inch by 7 foot compartments in line, in addition to one pipe compartment of 2 feet 6 inches by 7 feet (fig. 10). This shaft is about 5,100 feet deep and connects with the vein by crosscuts. It is used principally as a supply shaft for the mining areas at the north end of the property from the 67th

level down and also contributes largely to the pumping requirements of the property, having two sets of electrically driven, horizontal duplex plunger pumps, one at 1,000 feet below surface and the other at 2,600 feet down. These pumps handle the major portion of the surface water entering the mine at this end.

No. 12 Hecla containing one skip way, about 7 by 7 feet and a manway about 4 by 7 feet, was sunk in the vein at the extreme southern end of the orebody. This shaft is used principally for raising and lowering men and as a supply and equipment handling shaft. It is bottomed on the 81st level and is now the only inclined shaft in operation from this level to surface. In all other inclined shafts, above the 81st, pillar-removal operations are in progress, retreating from the bottom toward the surface. The 81st level is considered to-day the top of a new mine in process of development, and the extensions of two slope shafts below this point are separately designated as "A" and "C" slopes (fig. 2). In line with the cross-cut from No. 3 Tamarack to the vein, a third slope shaft is started below the 87th level. This slope is connected with "C" shaft and served by the 81st level haulage to Red Jacket shaft. Thus all parts of the property are connected to the 81st haulage way, and it may be well to describe this project a little further.

Eighty-First Level Haulage Way

As mining progressed to depths between 7,000 and 8,000 feet on the vein (4,900 feet vertically), the crushing load concentrated upon the inclined shaft pillars became such as to cause frequent interruption to hoisting operations. Large repair crews were required at all shafts, and maintenance work was necessary during the hoisting shift as well as between shifts. In the case of the Conglomerate mine, when ground movement began, such tremendous weight of hanging-wall material was involved that pressures reaching the "rock-burst" point were encountered. The exceedingly hard and abrasive conglomerate lies directly upon a highly brecciated lava top or amygdaloid, the amygdaloidal fragments containing considerable calcite, chlorite, and other soft minerals, while the material cementing the fragments consists of a very friable rock of fine texture, much resembling volcanic ash. The thrust on the Conglomerate mine shaft pillars is to a great extent absorbed by this less resistant foot material, which loses shape during the "squeeze" and bulges into the shaft on the footwall side, causing the worst form of shaft disalignment (fig. 11). Because adequate protection is impossible against a movement of such intensity, and because there is such a tremendous investment tied up in the millions of tons of shaft pillars, the management was prompted to study possible ways by which some plan of operations might be found which would permit realization upon these pillar reserves, would reduce shaft maintenance charges, and at the same time would assure the ability to prosecute mining to still greater depths unhampered by limited and uncertain shaft capacity. The proposition of sinking a vertical shaft in the South Hecla area similar to the Red Jacket shaft was unattractive because of the trend of mineralization toward the north and the consequent narrowing down laterally within the property lines.

As a result the 81st-level project was undertaken after acquiring the Tamarack lands, which occupy the center of the Calumet conglomerate area, as shown on the map attached (fig. 2). It may be considered, in a way, a reestablishment of surface facilities at a depth below surface of 8,100 feet down the dip. In principle, it is a scheme for concentrating at the 81st level all main haulage requirements for a mine below this level, locating on the 81st the collars of all shafts penetrating below this point, and making possible the ultimate use of existing vertical shafts for rock hoisting and other purposes, so that continual,

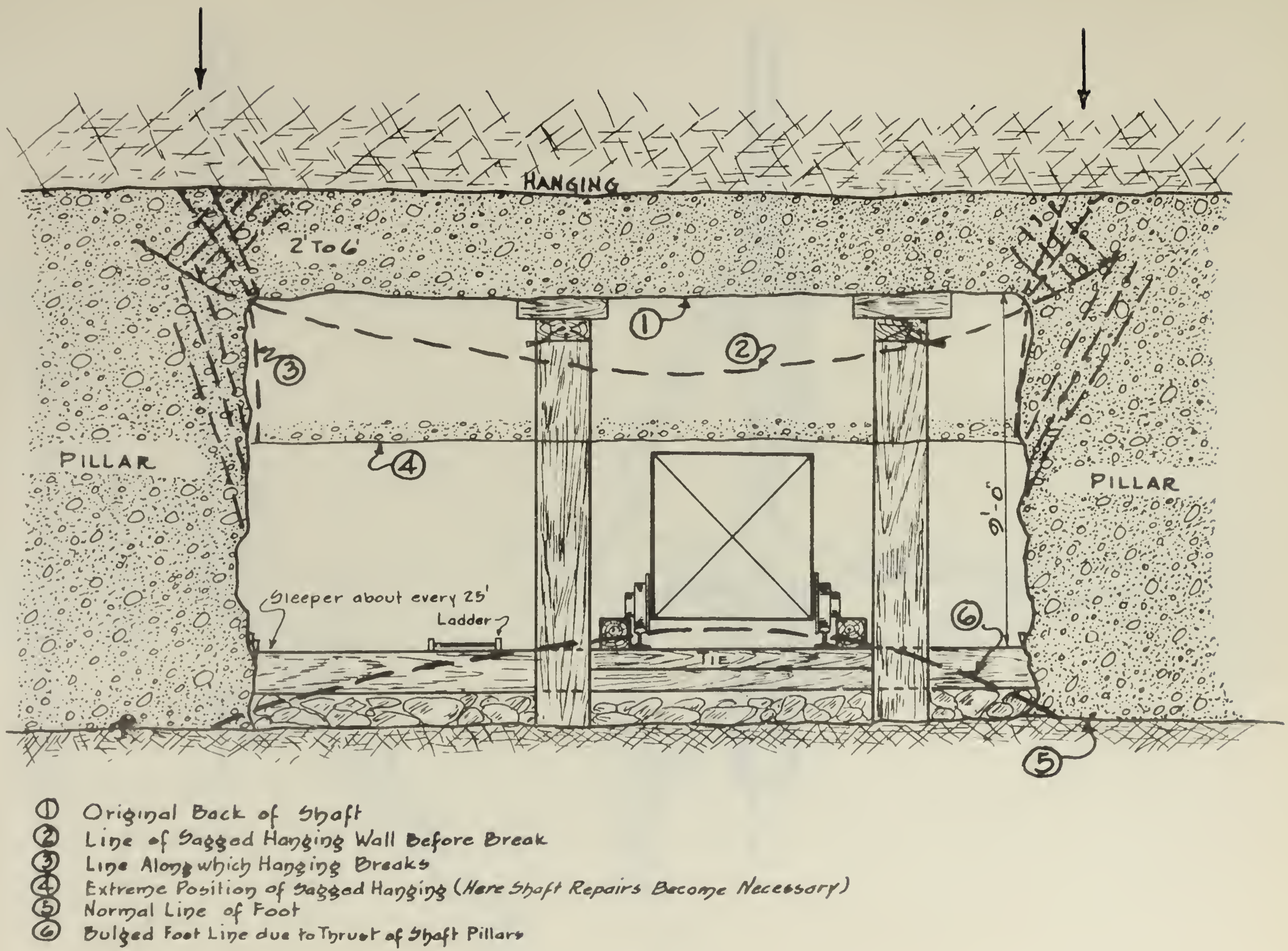


Figure 11.– Typical shaft cross section showing lines along which hanging and foot bends take place

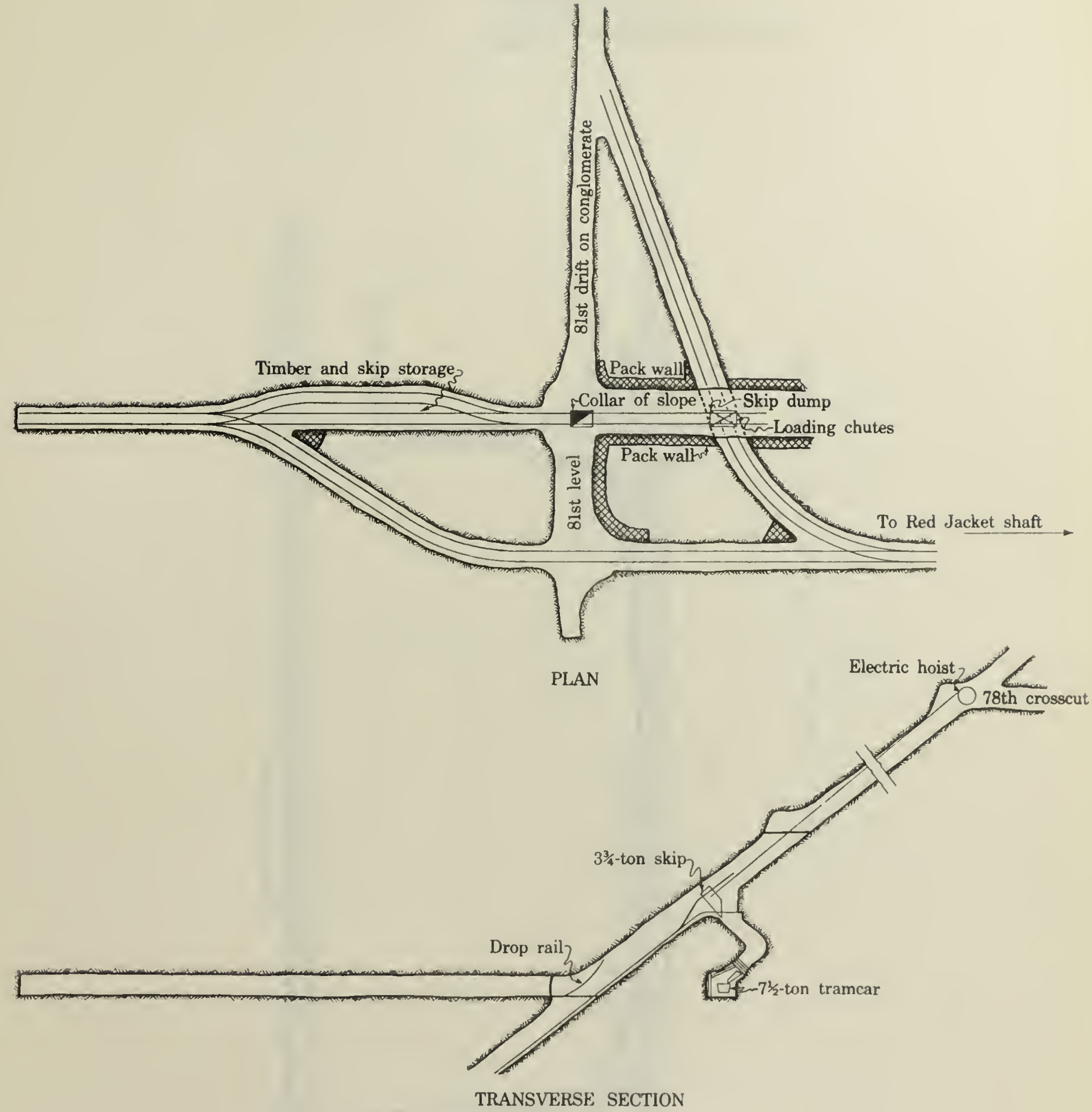


Figure 12.—Collar layout at "C" shaft, showing drop rails

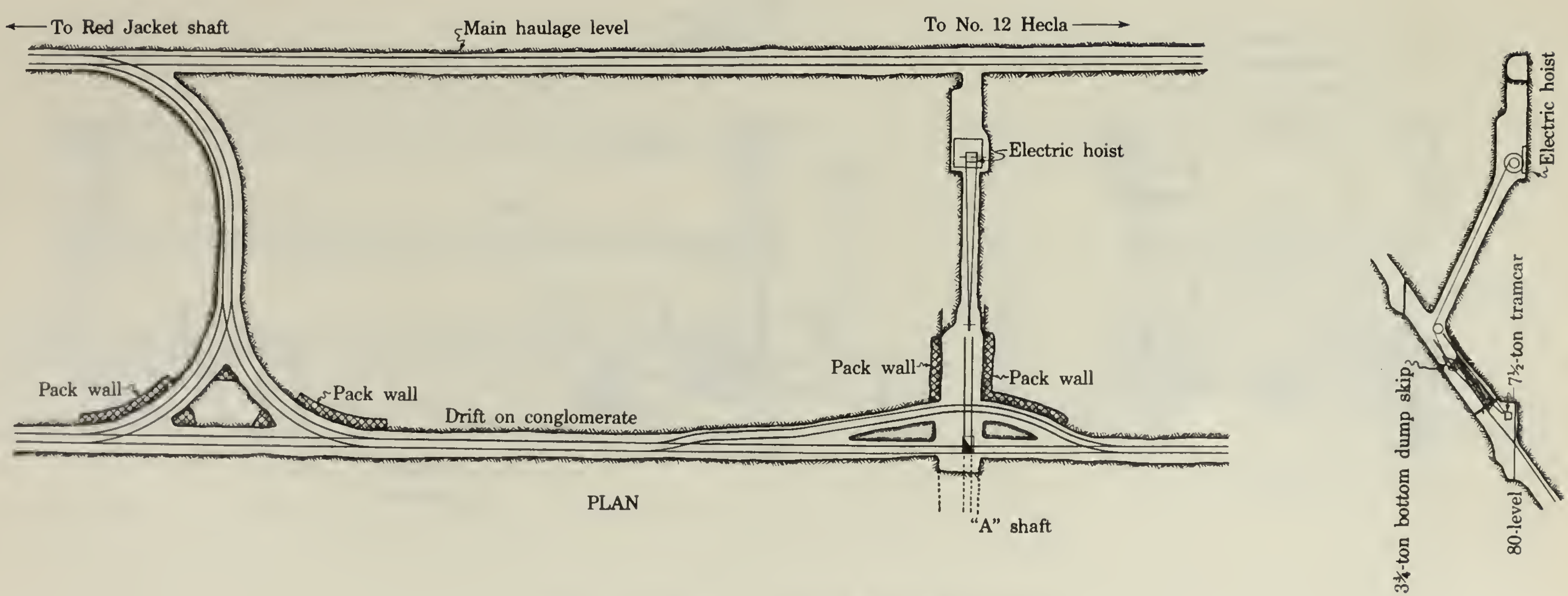


Figure 13.—Collar layout at "A" shaft, showing electric hoist

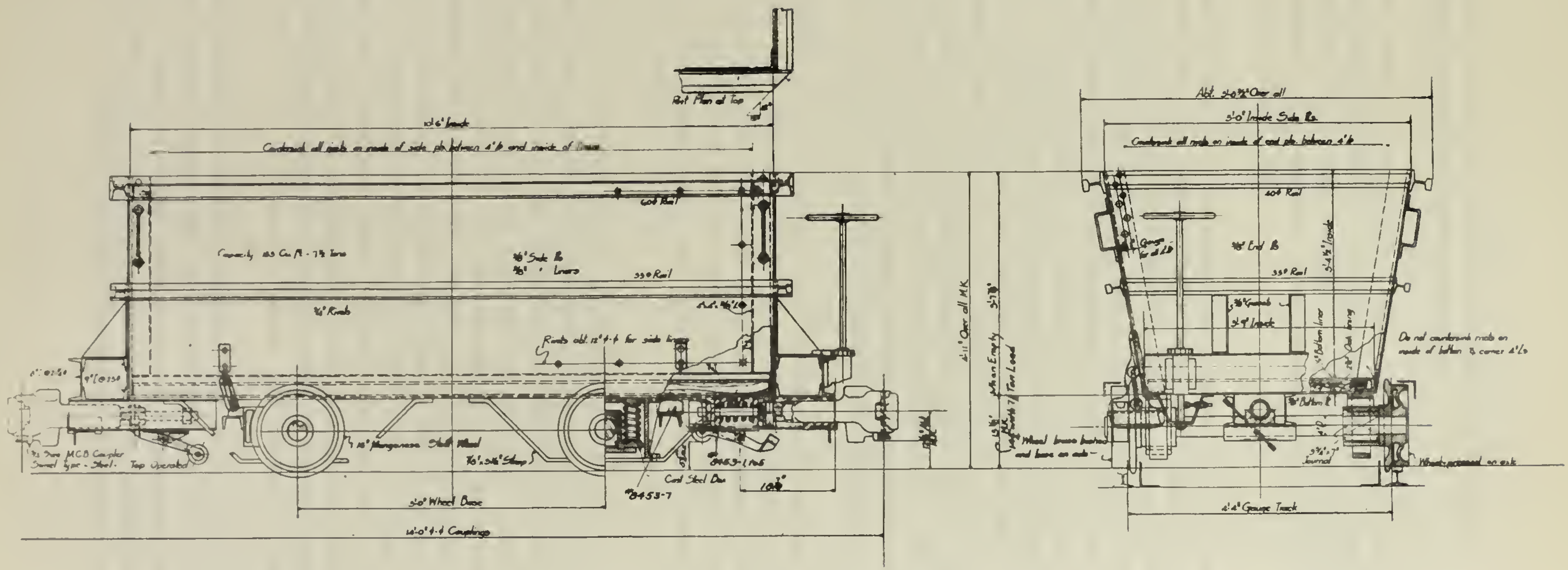


Figure 14.— Tramcar, 7½-ton capacity

full-capacity production may be maintained at minimum cost up to the time in the future when general "scramming" operations become necessary.

To meet these requirements a through-going tunnel paralleling the conglomerate bed which would give direct access from the Red Jacket shaft to the Hecla and South Hecla workings at the south end was deemed necessary. To fulfill the requirements of safety, permanence, and low upkeep, the selection of a location with reference to the Conglomerate mine workings was of vital importance.

Although failure of the hanging wall rock ordinarily decreases more or less regularly with distance from the conglomerate, movement of hanging in mass results when the formation is cut by a large number of prominent slips along which the movement may be distributed, with the result that the live load is carried to considerable distances above the workings. The footwall was, therefore, considered to offer the only stable ground, and it was decided to drive the tunnel along an amygdaloidal bed which lies 200 feet horizontally below the conglomerate. This bed was found to consist of a soft amygealoid of the fragmental type, barren of copper mineralization, and to be of sufficient width (approximately 30 feet) to encompass the entire cross section of the tunnel (9 by 12 feet) and still leave enough of the bed material overhead so that breaking into the base trap of the overlying flow might be avoided, and the probability of block caving thus averted. The distance below the conglomerate to this bed is sufficiently great to permit absorption by the surrounding traps, of the major pressure resulting from the closing in of stoped-out areas above.

The length of the haulage way, including the enlarged Red Jacket crosscut, is 9,800 feet from the Red Jacket shaft to the south extremity at No. 12 shaft. To facilitate handling timber, drill steel, and other supplies and equipment from this shaft to the haulage way and thence to the incline shafts below, all track gages have been made the same (4 feet 4 inches), and direct connections have been made between shafts and haulage by means of curved "drop rails" (see fig. 12).

The "A" and "C" shafts are equipped with electric hoists located on the 81st level, capable of winding from a depth of 3,000 feet below (see fig. 13). Haulage equipment consists of a motor-generator set with capacity for two 13-ton and one 10-ton General Electric trolley locomotives operating from a 250-volt d.c. trolley. Track is of 50-pound rails laid to a 4 foot 4 inch gage. Tram cars of 7½-ton capacity (fig. 14), solid-body type, and trains of 12 cars are loaded from pockets at the incline shaft heads, and dumped into pockets at the Red Jacket shaft by means of air-operated rotary dump cradles. Locomotives and cars are connected by M. C. B. three-quarter size couplings with tripping device by which single cars may be rotated without breaking train. Seven and one-half ton capacity Kimberley skips operate in balance through the Red Jacket shaft. All trains operate under control of a complete automatic, electric block-signal system, and to control the mine ventilation all doors on the main haulage way are mechanically opened and closed by trains in motion.

Although mining has already extended well below the 81st level and the increasing effect of pressure has had ample time to make itself felt in the immediate vicinity of the haulage, to date no timber or other support has been necessary over a great portion of the haulage opening. The entire length has been given a coating of "gunite" to prevent air leaching of the softer minerals, but in a few places timbering has been necessary.

DEVELOPMENT BY DRIFTS

Development of the vein from the incline shafts consists in cutting plats (fig. 5) or stations at each 100 feet of incline depth, from which 7 by 8 foot drifts (fig. 6) are driven to the limit of the adjacent mining area - that is, to the property limit or to a pre-established limit between shafts. The timing of drift advance is such that the upper drift of a group shall reach the mining limit before the top stope of a tier of retreating stopes has reached the shaft pillar. This permits the top stopers to move down and start a new stope at the mining limit without loss of time.

RETREATING STOPING

The length of stopes has been consistently shortened as depth was attained. In the original mining operations, stulls were used entirely for timbering stopes; later, due to the increase in vein thickness and quality, square-set timbering was used in all the stoping, and the stoping length was 200 feet. This square-setting was standard practice down to a depth of 5,500 feet along the lode; but as greater depth was attained, and as it developed that the number of stopes running over 20 feet in thickness was rapidly diminishing, and also in order to save the excessive cost of the framed square-sets, a gradual transition was made stull timbering, which is now used entirely for temporary support in all stopes. The stoping length was at the same time cut down to 100 feet. Timbering the lower portion of stopes has become largely a matter of standard practice, but the best judgment of the operating foreman and head timberman is consulted to determine the number and the spacing between stulls set in the upper portion of a stope. Standard practice at the present depths calls for four rows of stulls, in clusters of two timbers alongside each other (fig. 3) and 18 inches to 2 feet in diameter, set with approximately a 7-foot clear space between the near timbers (in the row measured horizontally), and about 7-foot centers between rows measured along the dip of vein (see sketch of stope, fig. 3). These double batteries cover approximately 40 per cent of the height of the entire stope and the practice of placing the double sets never varies up to that minimum height, regardless of how firm the hanging may appear during the early days of the stoping operations. Above this height (about 40 feet) the stulls are placed singly with a variable increase in distance between centers, the actual spacing being governed by the appearance and action of the hanging in each particular case.

Experience has shown that during the time required to complete a stope (approximately 100 days) the maximum thrust of the hanging wall manifests itself at a point about one-third the total height of the completed stope, or just below the fourth double set above the level. This is considered the critical support point in the entire operation, for while it is true that the three double sets below must carry their particular load for a longer period of time, the actual thrust upon them is considerably reduced, due to the proximity of the unmined ground immediately below the level. Set 4 is just below the center of the stope height, 40 feet above the level. It must hold the back for about 60 days after being placed, whereas the next set above (the first single stull to be placed) is well toward the center of stope height and thus above the point of maximum thrust of the vertically imposed hanging load, also the time during which the pressure acts upon it is considerably shorter. Each successive set from this point up, is required for a shorter length of time, and is located under an area of hanging wall which receives great relief of pressure due to the support afforded by the caved-in material of the stope immediately above. Frequently the last 20 to 25 feet require no timber, or at most a few odd stulls placed arbitrarily.

In drilling and breaking the vein at the stope face, three machines and some times four are operated simultaneously in each stope, the number depending upon the speed of extraction

deemed necessary on account of the pressure conditions in the particular stope in question. The operation may be classed as an "under-slicing" or "bottom-slicing" plan, each slice taking about 6 feet along the entire 100-foot length of the stope. The first slice for a new stope, known as the "cutting-out" slice, is started at the end of the proposed stope nearest the shaft and is extended toward the previously completed stope, which is usually well caved down by the time this first slice reaches it. Each successive slice thereafter is started from the side of the caved stope and extended toward the shaft (fig. 4), thus avoiding the short raises which would be necessary if each slice were started from the shaft side, and also avoiding the dangerous projection which would exist at the caved-stope end of each slice, just before the slice was completed. It is a generally accepted fact here, and it is believed throughout the entire deep-mining world, that such projections present conditions most favorable to rock burst.

It may be well to describe here the manner in which the miner is protected from danger while starting the slice at the edge of the caved stope. Just before each stope is completed to the level above, a double row of stulls known as the "breaker row" is placed from bottom to top, within about 3 feet of the shaft side or within 3 feet of the ground next to be stoped (fig. 3). The row consists of pairs of heavy logs, each at least 2 feet in diameter, set with a space of 3 feet between pairs (measured along the dip), thus leaving a safe area 3 feet wide, into which the hanging may not cave until the new stope has been sliced off. This row is not placed earlier during the progress of stoping, for the reasons that approximately 3 months is required to complete each stope and that the average life of the timber used, principally hemlock, maple, and birch, is from 4 to 6 months. Therefore from the time of completing one stope to that of completing the next, the major portion of the useful life of the "breaker row" has been expended, which is the end to be desired. As the last slice at the top of each stope is cut, the floor arch of the level above is drilled, preparatory to blasting (figs. 3 and 7), but the actual breaking is deferred until the slice has advanced sufficiently to leave a short length of the protecting arch above the driller. The stope above is by this time completely crushed in, but the danger of a sudden rush of caved material into the stope below has been obviated by a wall of heavy lagging timber, laid horizontally along the upper side of the "cutting-out" stull row at the time of completing the upper stope (fig. 3). This lagging is placed as the last timber operation in all stopes, anticipating the recovery of the floor arch from below.

The successive recovery of each stope on a particular level is a retreating operation from a preestablished boundary between adjacent shafts, toward the extraction shaft for the area in question. A complete mining operation on either side of a producing shaft is a series of four retreating stopes on four successive levels, the top stope leading the next below by approximately 150 feet, and so on down (fig. 4). This lead is considered a minimum distance over which the floor arch should remain intact during the active life of the stope above, and it also permits a lapse of time more than sufficient to consume the useful life of the timber supports in a particular stope before the floor arch removal operation reaches it, thus relieving the pressure on the arch at the time of removal.

No attempt is made to speed hanging collapse in stopes by blasting out or otherwise attempting to remove timber supports. Any regulation of the time of collapse must be made by increasing or decreasing the number of stulls placed during the active stoping operation.

As mentioned previously, experience of the past has demonstrated that points at which so-called "air blasts" or "rock bursts" are more likely to occur are at projections or points of rock, such as intersections of drifts and crosscuts, turnouts, entries to engine

rooms, projecting points on stope faces, etc. It is unavoidable that such projections must be made in laying out a complete haulage system, together with space for motor generators, transformers, fans, pump rooms, etc. In deep mines where pressure of the overhanging load reaches rock-burst proportions, it is also unavoidable that such points of projection must be maintained to keep at a minimum the span of unsupported back in such places as must be kept open for an extended length of time. Timber sets of long-life material such as fir or pine are of prohibitive cost and require frequent replacement. The support which has given the greatest satisfaction on the Calumet conglomerate lode, in life, cost, and safety, consists of a "pack wall" or built-up block of "dry wall" with interbedded layers of flat timber, the timber laid horizontally and built up with the broken rock; about one-sixth of the total cubic volume of a block consists of timber. The first cost of construction of this "pack wall" for a given area of hanging is the same as that of supporting a similar area by means of stulls of short-lived timber, such as in stopes. The labor used in placing the rock-timber block is double that for placing stull timber, but the cost of material offsets the high labor charge. Where this form of support is used, the point of rock is first removed entirely and the rock-timber "crib" is built up in its place. As the "squeeze" takes place over such cribs, the rock is forced tightly against and into the timber layers, cutting off circulation of mine air and materially preventing timber decay, with the result that the useful life of the block is several times that of any timber support which might be used. In maintaining the sides of inclined subshaft heads where severe slabbing has taken place, such cribs have been built up in the form of walls from 6 to 10 feet thick, along the dip of the shafts (about 36°) and have proved of inestimable value, due to their ability to cushion the thrust and compress very slowly, giving ample warning of crushing pressure on the timber of the shaft-way with the result that time is allowed to replace crushed headers on the shaft sets before the sets themselves have been ruptured.

Under the longwall retreating system adopted for the deep Conglomerate workings, bursts have become quite a rare occurrence and are not looked for except in the last or next to the last stope at the shaft pillar. In removing the shaft pillar, although the adjacent stopes have by that time caved tightly, it may be said that almost every form of pressure result is encountered, such as wedging or the breaking of large masses from the pillars, heaving of the foot, back subsidence, and excessive drag. Pillar removal practice is indicated by Figure 8.

METHODS OF SAMPLING

No sampling of ore in place or in muck piles is practiced in the copper country of Michigan because of the difficulty of sampling ores containing copper in the metallic form in such irregular distribution and in pieces of considerable size. It is not believed that results approaching reasonable accuracy could be attained by sampling either the amygdaloid or conglomerate lodes of this company. In formations such as the porphyries where the copper occurs in minerals of brittle or earthy character, sampling in place produces results of considerable accuracy.

The general concensus of opinion among mining men in the Lake Superior district is that mine sampling does not pay. Mill-feed sampling is not practiced generally, but in special cases may be used to make a rough estimate of grade of ore from a certain section.

In past years this question has continually turned up for discussion and decision, and on a great many occasions the attempt has been made to prove the practicability of sampling, with questionable results at all times, and in some particular cases with results so misleading as to constitute a menace rather than a help. Consistent diamond drilling into the footwall at regular short intervals, in the case of a mine where copper was known to

occur in very profitable patches in the footwall, proved very expensive to one company because of the misleading character of the information.

The best method of determining what is and what is not ore is to keep the section foreman in constant touch with the assay value of the mill product coming from his branch, by written reports to him. He then possesses a daily knowledge of what all stoping faces show in the way of exposed native copper and its continuity and degree of concentration; this data, combined with his report of mill results, gives him a criterion for judging the grade of ground of a certain appearance.

On his daily trips into development openings, he notes the condition of the face and estimates its grade. At the end of the month, he combines his notes of daily estimates and arrives at a composite estimate of grade developed by the month's work. Maps are made up to show the estimates by months and are checked against mill returns when this area contributes ore to the mill. These checks are, of course, quite general and must be made by engineers familiar with field conditions.

Owing to the great linear extent and depth of the operating mine and the care with which past records have been kept, it is possible to gage (without sampling) the grade of ore now exposed and also that to be developed in the immediate future with a degree of accuracy not to be attained by sample assays.

Estimation of Tonnage and Value in Mines of Calumet and Hecla

Estimates of tonnage extending beyond the limits of complete development and the grade assigned to such extensions are carefully and conservatively computed and have been subjected to repeated check by actual mill extraction records over a period long enough to have brought the procedure to a state of high reliability. Each step is based on conservative estimates before it is introduced into the combined computation. All geological indications are carefully considered, and only such extensions of the ore lines are made as can be safely counted upon for the future on the basis of the obvious tendencies of the vein at the time.

The thickness and perimeter of the stoped areas are carefully recorded and mapped. The exact area of vein mined over is scaled from the maps, and with a knowledge of the mill returns for various areas, which are separately concentrated, computations are made showing the copper content of the ore coming from a particular area and also the number of pounds of copper per unit of area cleaned up. In the development openings, which are continually being advanced at greater depths on the lode than the average depth of stoping operations, frequent crosscuts are made to hanging and footwalls, to determine in advance of stoping operations any tendency of the workable vein to widen of pinch. Mineralization for a particular vein may show a tendency to improve or decrease as greater depth is attained, due either to a change in the form of distribution throughout a given area or to a change in the thickness of vein throughout which the same form of mineralization may be distributed. Such changes are usually not precipitate, however, and lacking the sudden appearance of a major fault, cutting off vein mineralization, the tonnage and grade which may reasonably be expected can be estimated with considerable accuracy for a number of years in advance of actual operations The probability of the sudden appearance of major faults, such as mentioned, can be anticipated with a fair degree of assurance by a careful study of any unusual undulations appearing on the survey maps of advanced development openings. The long record of the district as a whole indicates that such occurrences are very unusual and are less likely to appear as depth is attained.

Ventilation

Due to the high rock temperature in the lower portion of the mine and the relatively low average surface temperature, all primary ventilation is accomplished entirely through the medium of natural air currents.

From the records of the United States Weather Bureau, covering a period of 20 years the average mean surface air temperature is 40.5° F., dry bulb, the average mean relative humidity for the same period being 82 per cent, corresponding to an average mean wet-bulb temperature of 38.2° F. At a vertical depth of approximately 5,502 feet virgin-rock temperature measurements recently taken show a temperature of 93.9° F. on the 93rd level. This indicates a rock temperature gradient of 1° rise for each 103 feet of vertical depth from surface. The gradient from 4,800 to 5,500 feet vertically (81st to 93rd levels) by the most recent tests appears to be 1° for each 100 feet. All temperatures were obtained by placing three standardized mercury thermometers, calibrated to tenths of a degree, in holes 10 feet deep drilled for the purpose in the side walls of advancing development openings as closely as possible to the advancing face. The collars of the holes were blocked to air currents while thermometers were in place, and periodic readings were made on all three over a sufficient period of time to avoid the possibility of including any effect of the heat of blasting in the finally accepted virgin-rock temperature. In some cases in order to gage the possible effect of air cooling due to the exhaust of drilling machines, the holes were drilled at the end of a short crosscut, which was bratticed off from the advancing development opening.

The great difference in rock temperature between surface and mine workings produces a difference in air densities in different parts of the mine, causing a natural air flow, the major currents of which remain fairly constant in direction throughout the year, because of the comparatively small effect that changes in surface air temperature have upon the average temperature of air columns of such great length. No great difference of collar elevation exists for any of the shafts throughout the mine. It is particularly noticeable in natural-draft mines that the larger volumes of fresh air tend to go directly to the lowest openings, due to the rarifying power of the high-temperature zone. In the ventilation of the upper workings this is largely compensated for by large openings between shafts, such as drift stopes and stoped areas, presenting low resistance to the flow of air. As all stopes retreat from boundaries toward the slope shafts, extracting floor pillars as they go, a large volume of air circulates at very low velocity through the caved material of the stopes. The airways of the mine, as compared with the average deep mine, are large and kept free from obstructions. All shaft timber is placed in good alignment and drifts and crosscuts are usually free of timber. Where openings are kept up by "pack walls," the wall face is well aligned and smooth. The following table gives the analysis of mine-air samples taken by G. E. McElroy of the United States Bureau of Mines and shows the air to be of good quality throughout the mine.

Air-sample analyses - Conglomerate mine of the Calumet & Hecla Consolidated Copper Co

Bottle No.	Laboratory No.	Date, 1928	Time, a.m.	Location and description	Volume of air, c.f.m.	Chemical Analyses			
						CO_2	O_2	N_2	CO
381	48,537	7/11	9:10	Return air from Red Jacket slope section on 57th level	7,000	0.15	20.80	79.05	0.00
383	48,536	7/11	9:50	Face of 58 S. stope off Red Jacket slope	2,500	.11	20.76	79.13	.00
380	48,534	7/11	11:15	Face of 64 N. (abandoned) drift off Red Jacket slope	None	.22	20.36	79.42	.00
379	48,535	7/12	11:35	Face of 92 S. drift 100 feet off "C" slope	Drilling exhaust	.10	20.85	79.05	.00
391	48,538	7/12	11:50	Face of 91 S. drift 700 feet off "C" slope	Drilling exhaust	.12	20.83	79.05	.00
390	48,539	7/13	12:30 p.m.	Return air from stopes, on 80 drift 2,000 feet north of No. 6 Hecla shaft	7,000	.09	20.78	79.13	.00
385	48,591	7/14	10:00	No. 5 Tamarack upcast shaft at 29th (71 R.J.) level	37,000	.05	20.92	79.03	.00

Every effort is made to keep all downcast shafts as dry as possible in order to reduce the relative humidity of the air at the working places.

At a vertical depth of 4,750 feet, at one of the downcast points, dry-bulb temperature is estimated to be about 14° below virgin-rock temperature for this depth. Measurements at this depth at No. 12 downcast, show 64° wet-bulb and 72.5° dry-bulb temperature, corresponding to 63 per cent relative humidity. For such a depth this is considered remarkably cool, dry air and is considered to be due in part to the dry condition of the shaft, but to a still greater degree due to the fact that the mine water contains a high content of natural calcium chloride, which materially reduces its rate of evaporation.

Auxiliary ventilation of development headings and dead ends is taken care of by electrically driven compressed-air blowers. Fan-pipe installations driving large volumes through short distances are used in ventilating deep pump and hoist rooms.

In the two main pump rooms at No 5 Tamarack shaft, where a large volume of heat is generated by three 350-hp. motors in each room, single-inlet fans are used having 3-foot diameter rotors 1.5 feet wide, operated at 750 r.p.m. These each deliver 30,000 cubic feet of air per minute from the rear of the pump rooms to the shaft.

DRILLING

Machine drills of the Leyner type with 3½-inch pistons, mounted on 3-inch single jack posts are used entirely for stoping and drifting and are operated under 80-pound air pressure. Jackhammers are used only on special work, such as trimming in places and blockholing generally. Hollow hexagonal drill steel, 1 inch across flats, and with 1/4-inch holes is used throughout, a set consisting of a 2-foot starter and drills varying by 1-foot lengths from 3 to 10 feet, inclusive. Gage of bits is reduced 1/16-inch for each increase of 1 foot

in drill length; the starter bit is of 1 3/4 inch size and the 10-foot finisher is of 1 5/16-inch gage. Crossbits are used entirely because of the tendency of single bits to jamb on account of the cracks encountered at the face in the deep Conglomerate mine workings. The placing of holes in stoping slices is left largely to the miner, supervised by the shift boss, and varies considerably because of the changeable nature of the face due to pressure slabbing or to jointing. In drifting, the pyramidal drawcut is standard practice, a cut requiring usually 18 to 20 holes, 6 feet deep, to give an advance of 4 1/2 feet per round (fig. 6).

BLASTING

All blasting is done at the end of the working shift. Powder is delivered to each miner once a week and stored in metal boxes provided with locks. Caps and fuse are kept in separate metal containers at some distance from the powder. The powder sticks are 1-1/8 inches in diameter, 8 inches long, and consist of 30 per cent nitroglycerin for old backs, 40 per cent nitroglycerin for shaft pillars and stopes, and 70 per cent giant gelatin for drifts. In charging, the primer is usually set near the bottom of the charge and firing is done by fuse and detonators. Timing of firing is determined by the length of fuse cut for the particular hole. For special work, such as long raises and vertical shaft sinking, electric delay blasting is used with great success.

TRAMMING

The tendency of the conglomerate rock when blasted is to break into slab-shaped pieces, so that in spite of the relatively flat vein dip, the greater portion of the material broken in stopes runs to the bottom. In shaft-pillar removal, timber chutes are built at the stope bottoms for car loading, but ordinarily for the retreating stopes rock is scraped from the drift floor up a scraper incline of sufficient height to permit the tramcars to be run underneath and filled by ore dropping through an opening in the scraper platform above (fig. 15). About 15 per cent of the stope production coming from the upper stope faces is scraped down to the level by the same scraper that is used to fill the tramcars on the level (fig. 16).

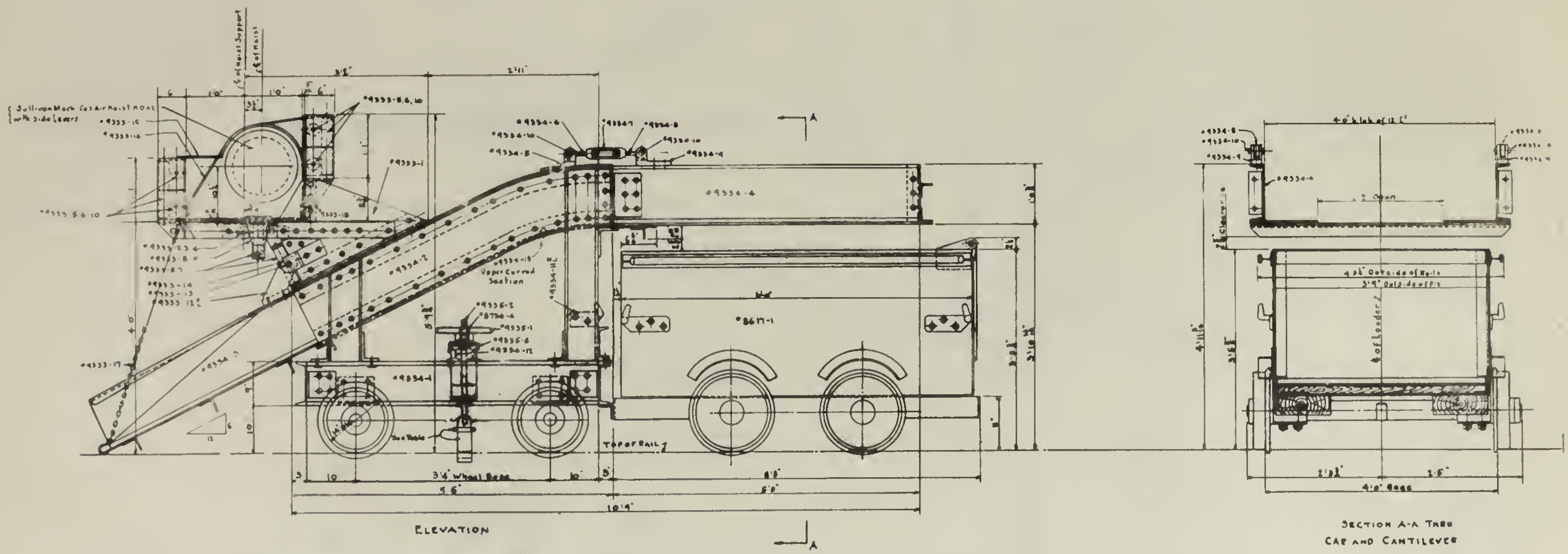
To accomplish this, a "snubbing-post" and head pulley are set at the top of the stope, and the scraper drag line after passing through a pulley on the hanging-wall side of the level is diverted to the head pulley. After cleaning out the stope, it is a simple matter to replace the head pulley on an inside stull and resume scraping along the level.

All mine cars employed in development work and stoping are of 3-3/4-ton capacity and of the end-dump type, having 4-foot gage. The haulage, except on the 81st level, is by storage-battery locomotives, two types being used: 3-ton, single-motor, 48-cell MV13 Exide batteries; and 7-ton, single-motor, 48-cell MV27 Exide batteries. Each of these are single-wheel drive through double-reduction gears.

The motor-generator sets are of two types: 40-kilowatt, 125-volt, d.c., driven by 60-hp. motor; 20-kilowatt, 125-volt, d.c., driven by 31-hp. motor.

PUMPING

Glacial drift running from 10 to 40 feet in depth covers the Calumet area, and while it is not possible to drain this deposit thoroughly over the entire area above the mine



Rails	25lb	35lb	40lb
Rail Clamps	#9338-7	#9338-4	#9338-5
Rail Clamp Ring	-8	-9	-9

Figure 15.— Arrangement of scraper slide

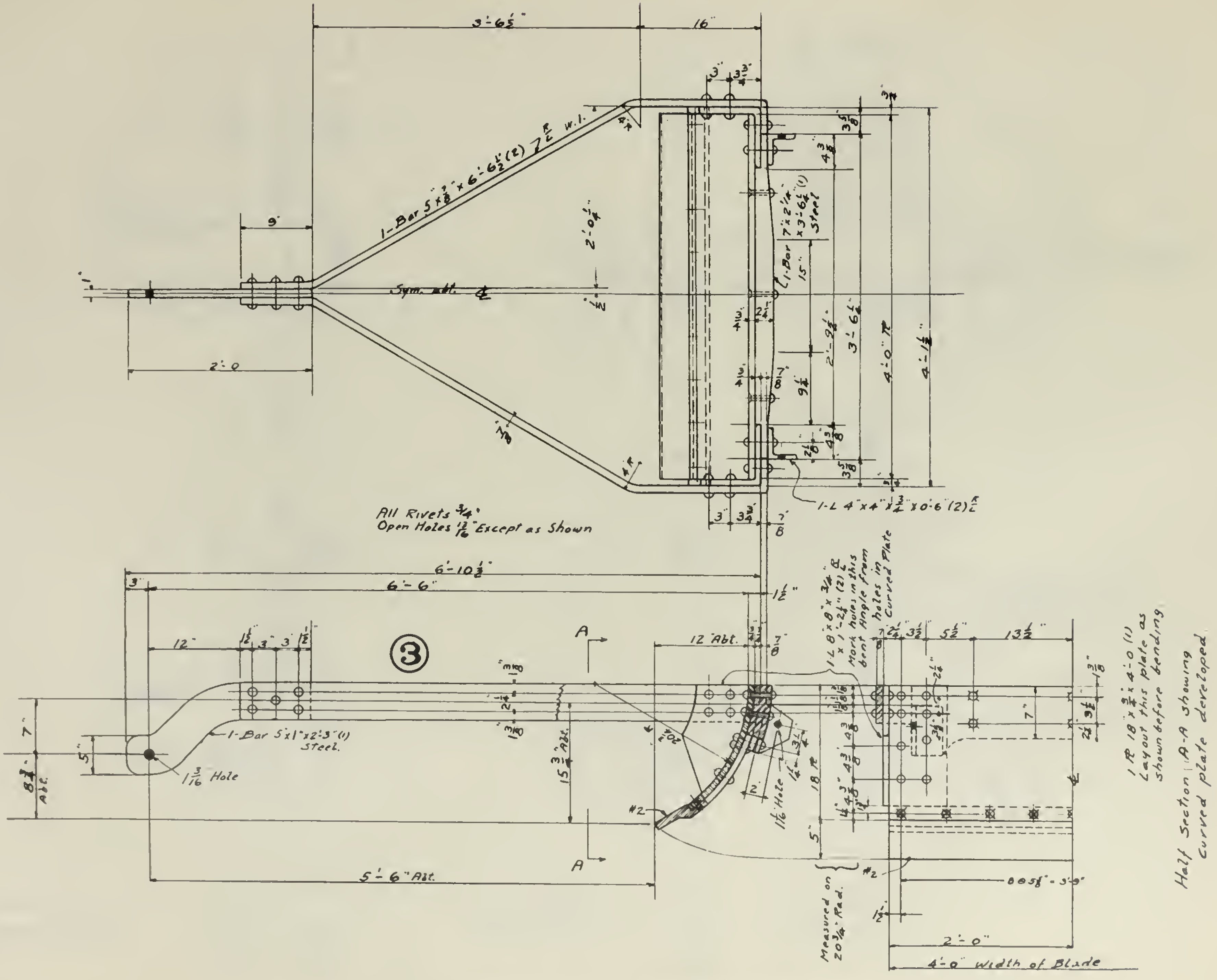


Figure 16.— Stope and drift scraper

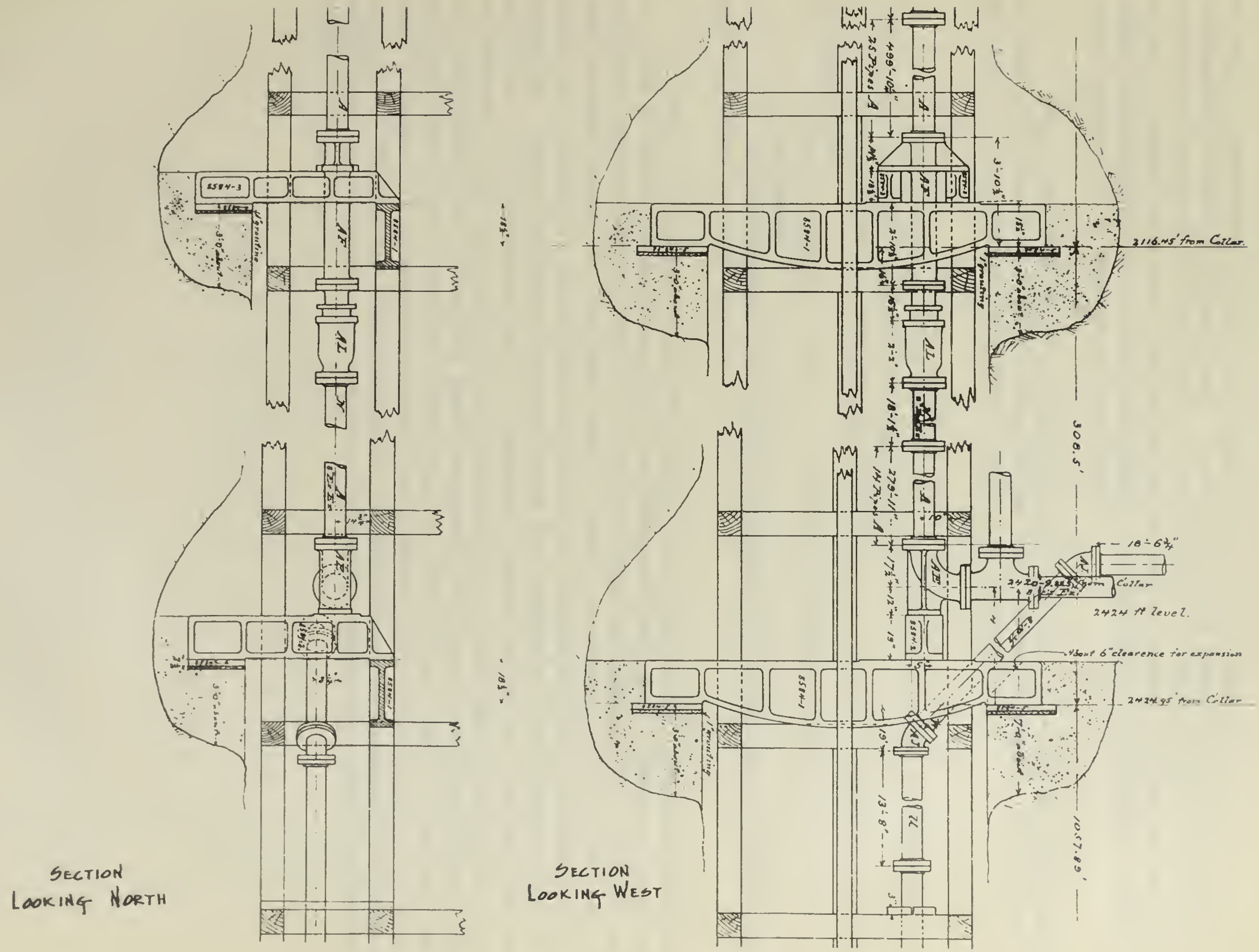


Figure 17.– Column pipe and supports at No. 5 Tamarack

workings, a considerable saving in mine pumping has been accomplished by the use of surface intercepting ditches and surface pumps. So-called "collar" pumps are placed at points nearest under the shaft collars where sufficient water can be caught to justify a separate pumping unit. Somewhat more than 2,250,000 gallons per day are pumped to surface by the combined pumping facilities in the Calumet conglomerate lode workings, the principal installation being located at No. 5 Tamarack vertical shaft.

This plant consists of two pump stations, one 2,424 feet vertically below surface, the other 5,156 feet down or 2,732 feet below the upper station to which it delivers water in one lift. At each station the equipment consists of the following:

3, 334-g.p.m. vertical triplex plunger pumps.

3, 350-hp motors, using 440-volt a.c., starting on by-pass accomplished by one-half voltage or 220 volts taken from transformers.

1, Air compressor, triple-expansion, delivering air to air chambers at 1,250 pounds per square inch.

3, Oil transformers, 500-k.v.a., step down from 4,000 to 440 volts.

1, Switchboard.

1, Fan, single-inlet, rotors 3-foot diameter by 1.5 feet wide, 30,000 cubic-foot capacity, driven by 75-hp. motor.

Water is supplied to the lower pumps from a large-volume storage sump and relayed to a sump of nominal capacity just off the 2424-level pump room. The shaft water column consists of 8-inch diameter steel pipe, heavily galvanized inside and out. Flange connections are used having male and female joints. During construction all pipes were threaded, flanges were screwed on, flange bases were welded to the pipe and then the welding strip was galvanized. Thus no threads or bare steel were left exposed to the corrosive action of the mine water. All pipes except short "filling-in" lengths are 20 feet long and the columns are supported in the shaft by heavy cast-steel girders, at intervals of about 300 feet (fig. 17). The girders rest on concrete piers, carried in hitches cut into shaft walls. Immediately below each girder, expansion joints (fig. 18) have been provided for the pipe line, to care for changes in pipe length which might occur during periods of interruption to the pumping service. The pipe and flanges used are of various weights from standard to double-extra heavy, according to the position occupied in the column. Details of supports and joints are indicated in the figures.

Table 1.- Summary of costs in units of labor, power, and supplies.

Name or number of mine: Conglomerate Period covered: Year 1930.

Ore tonnage: Mined and hoisted, 872,834; development, 40,964; mining, 831,870.

Mining method: Open stopes with temporary stull supports; retreating system.

	Development	Mining (stoping)	Total
A. Labor (man-hours per ton):			
Breaking (drilling and blasting)	1.413	0.386	0.434
Timbering	.308	.481	.473
Shoveling	1.077	.397	.429
Haulage and hoisting	-	-	.248
Supervision	-	-	.138
General	-	-	.890
Total labor underground	2.798	1.264	2.612
Av. tons per man per shift	-	6.328	3.062
Labor, percentage of total cost	-	-	69.84
Av. tons per man-shift on surface properly chargeable to underground operation	-	-	7.23
B. Power and supplies:			
Explosives:			
Pounds per ton	-	-	.75
Kind and grade	-	-	30-40 and 70%
Timber, linear ft.	-	-	6.192
Power, kw.h. per ton:			
(1) Air compression	-	-	4.942
Other supplies in percentage of total supplies and power	-	-	38.32
Supplies and power, percentage of total cost	-	-	30.16

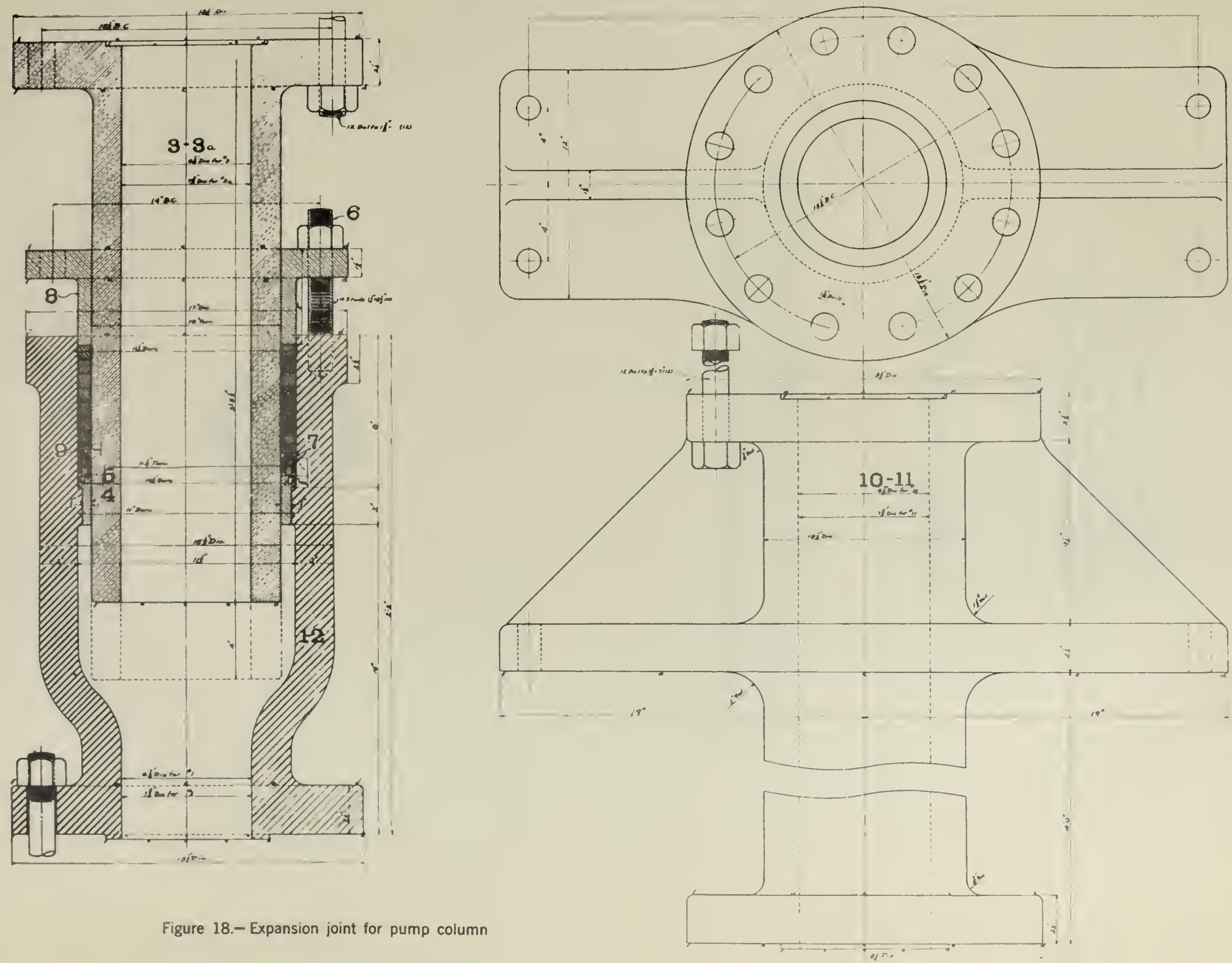


Figure 18.— Expansion joint for pump column

I. C. 6527

November, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

PRACTICAL RULES FOR THE USE OF THE MAGNETOMETER IN GEOPHYSICAL PROSPECTING

BY

W. Ayvazoglou

(Translated from the original French of
M. C. Alexanian)

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PRACTICAL RULES FOR THE USE OF THE MAGNETOMETER IN GEOPHYSICAL PROSPECTING[1,2]

Translated by W. Ayvazoglou[3] from the original French of M. C. Alexanian[4]

INTRODUCTION

A large number of organizations interested in the search for mineral deposits have adopted the magnetic method of geophysical prospecting, as it has proved to be the most simple, convenient, and rapid, and at the same time the least expensive in comparison with other methods.

According to the last statistical data, about two millions square kilometers were explored in the United States by different geophysical methods. Ninety-six per cent of this area was prospected for oil and 4 per cent for minerals. Detailed exploration by means of the magnetometer has been carried out over about 85 per cent of this surface.

The great advantages offered by this method resulted in the fact that the magnetometer was tried out in most places in which geophysical methods were applied. The main purpose of such exploration is to obtain the maximum knowledge valuable to the geologist; therefore good geological interpretation of the results of measurements is important. The possibility of drawing practical conclusions from a map representing the results of the study of the terrane depends greatly on the accuracy with which the measurements are made.

The purpose of this article is to show the different sources of error by which the work of an operator may be affected and to discuss the means by which the carrying out of a magnetic survey may be improved.

1 From Annales de l'office national des combustibles liquids, No. 4, Paris, 1930, pp. 677-702.
2 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6527."
3 Translator, U. S. Bureau of Mines.
4 Engineer geophysicist.

All the details and characteristic properties of a magnetometer must be studied thoroughly in advance. A simple knowledge of how to turn the crank of the instrument or to take readings from the scale is not sufficient. Several months of practice in the field are required until all the operations, which seem so simple, are understood thoroughly.

W. M. Barret describes a very characteristic case which is given below, owing to its special interest:

> Large expenditures were made by a petroleum company for the determination of a geological structure by means of a magnetometer. Several organizations were in charge of this work. In 1928 the author had an opportunity to examine the three magnetic maps showing the variation of the vertical component in the same region. The maps looked to be drawn very carefully. The comparison of the maps was easy, as all the measurement were made along the same profiles, drawn according to a regular distribution of the stations of measurement. However, a striking difference could be noted at the first glance. The difference became still greater after the results of measurements were reduced to the same base and the regional corrections were applied. Important magnetic anomalies indicated in one of the maps were entirely missing in the two others. The minimum shown in the last two maps was in the same region, represented as a distinct maximum in the first map. It was evident that this maximum was created solely by the operator. It was then discovered that the operator had worked on magnetic methods only for three months.

Similar cases could be established even with persons having greater experience in operating physical instruments. Many errors assigned to the magnetometers were in reality caused by the lack of ability of the operators, even of the most conscientious ones. To eliminate this disadvantage for others, the author describes in the subsequent paper the practice developed by him in his field work.

The principles of construction of magnetometers used in geophysical prospecting, known under the name of magnetic balances, are the same. The magnetometer used the most is that manufactured by Askania (for vertical as well as horizontal components) and next are the magnetometers constructed by Töpfer, Exploration, and Örtling. All these instruments are well elaborated and almost of the same order of sensitiveness.

Thus the quality of the work done by the operator depends exclusively on his ability to use the instrument. The operator must know thoroughly how the different constants, specified by the constructor in the bulletin of delivery, should be verified. These constants change usually during transporation on account of the shocks suffered by the instrument; sometimes (very rarely) the determination of the constants is not made with sufficient accuracy before starting on the expedition. Besides, the instrument is in general not adjusted according to the region in which it is to be used. Therefore the operator-geophysicist must know how to make all these adjustments.

SOURCES OF ERROR

The sources of error may be due to (1) construction, (2) observation, and (3) natural causes.

I. ERRORS DUE TO CONSTRUCTION

1. Variation of the Magnetic Moments of the System of Magnets and of the Bar Magnets

Even during a long period of use the variation remains insignificant under the condition that the magnetometer box is not opened too often. The magnets (lamellae) are manufactured from very good tungsten steel or chrome steel. The bar magnets, which serve as auxiliary compensation magnets, are from the same material. A box magnet with the magnetic moment equal to 961, as determined by the Askania Works in Berlin, was sent after one year to the same manufacturers for reexamination; the new value was established as being equal to 957; this proved that the variation was negligible.

This determination can also be carried out in the field, thus making the sending of the apparatus or the accessories to the manufacturers for the adjustment unnecessary. It is only necessary to have a standard bar magnet and keep it, well wrapped in cotton, in the laboratory away from ferromagnetic objects; the magnetic moments of the magnets used in the field can be determined by comparing them with the standard bar magnet (see the instructions issued on this subject by the manufacturers and laboratories that design such instruments). To keep these magnets in condition, the smallest shocks, humidity, and sudden changes of temperature should be avoided; the magnets must, besides, be kept away from ferromagnetic objects.

During the first six months the magnetic moment of these bars decreases appreciably, but after this time it remains almost without change; thus, the moment must often be verified during the first six months after the delivery of the bars.

The magnetic system in Haalck's variometer consists of two pairs of magnetized lamellae in the form of a cross; the ratio M_1/M_2 of the magnetic moments of these two pairs of lamellae must be determined at least every three months, as M_1 and M_2 never change in the same proportion; thus the value of the ratio changes in the course of time, although the lamellae are almost identical. The ratio M_1/M_2 varies usually from 0.85 to 0.95.

2. Displacement of the Center of Gravity of the Magnetic System

Displacement of the gravity center of the magnetic system usually results from shocks and temperature changes. The center of gravity is brought to its initial position by operating the two lateral screws secured to the cubical body of the magnetic system.

If the instrument is well adjusted for a fixed region the small displacements of the zero point, which may be established during the measurements, can be regulated by means of compensator magnets P; in this case the image of the scale must remain within the field without the interposition of these magnets; in other words, the lateral screws must be operated until the magnetic system of the instrument is almost horizontal or vertical (case of the balances assigned for the measurement of the vertical or horizontal component). The purpose of displacing of these screws is to displace the center of gravity along the abscissa.

A third screw, larger than the two mentioned above, secured beneath the cubical body, serves for adjusting the sensitiveness of the apparatus; by the operation of this screw the center of gravity of the magnetic system is displaced along the ordinate; the closer the center of gravity is to the axis of oscillation the more sensitive is the instrument. However, care must be taken that they do not coincide, as this may result in an unstable state of equilibrium of the system of oscillation. A useful compromise should be established between these two factors; the operator should not endeavor to secure a sensitiveness which can not be maintained by present-time instruments.

Being adjusted for a certain region the different constants of the apparatus must be determined at the place, as their values are necessary for the final calculations of the stations of measurement.

3. Displacement of the Zero Point of the Scale

The displacement of the zero point may result from shocks and the displacement of the magnets P. If the error caused by the displacement is greater than the experimental errors (± 10 gammas to ± 12 gammas), all the stations along the same magnetic profile at which measurements were made previously to the establishment of the error must be started over again.

This displacement is usually discovered after the return to the base station. The role of the latter is primordial, as not only all the following stations are connected with it but because it serves also for the discovery of the displacement of the zero point.

After having completed all corrections for the temperature and the diurnal variations, and in order to find out whether the readings taken in the beginning of the day work are the same, it is necessary to return to the base station three times during a working day -- in the beginning of the operations, in the middle, and at the end of them.

Therefore the place of the base station must be chosen so that it may be easily reached during the operations.

If the difference between two readings taken at the base station is great, this difference must be distributed either linearly among all the stations affected, or, if the operator wants to make sure that the results of measurements are accurate, the measurements must be made all over again. The application of the linear distribution of the error may be recommended for trained operators.

4. Determination of constants

Scale constants

a. Vertical Component.- In determining constants the following two methods may be used:

(1) the direct current frame method, and
(2) the method by using the magnets P (Gauss' principle).

1. By using the first method the coefficient of the self induction of the coil of balances manufactured by Askania, Töpfer etc., must be known. This is furnished by the manufacturers. Haalck's universal variometer does not require the use of the frame as, according to the theory of the apparatus, more simple, although less accurate, methods may be applied. The advantage of the use of the frame consists in obtaining great accuracy in the measurement of the scale constant; on the other hand, its transportation in the field is not sufficiently convenient.

2. This is why the method of using the magnets P must be preferred. In this case the operations are simple and easy; of course, the magnetic moment of the magnet used and its distance in millimeters from the axis of oscillation of the magnetic system of the instrument must be known.

By combining these two methods, the ratio M_1/M_2 of the magnetic moments of the two pairs of magnetized lamellae of Haalck's variometer can be determined with a sufficient accuracy; the operation to be carried out for this consists simply of eliminating the constants.

b. Horizontal Component.-

1. The case of simple balances (Askania, Töpfer, Örtling).- The operations are similar to those mentioned above, with the sole difference that the magnetic system oscillates in the north-south magnetic plane instead of the east-west. But the determination is more delicate. All the possible care must be taken for preserving the horizontal position of the instrument (the vertical position of the principal axis must be as perfect as possible). An inclination corresponding to one-half division in the level may cause an error in reading the scale equal to ± 10 gammas. In the case of the vertical component a difference in the level equal to almost one or even one and one-half divisions can be tolerated in the course of the measurements; the vertical position of the instrument measuring the horizontal component must be almost perfect.

2. Case of Haalck's variometer (Exploration).- The operations require much time and can be carried out only by using the auxiliary compensation magnets P and K. In addition to the horizontality of the instrument, care must be taken in turning its different parts exactly at 180° about the axes. Because the operations take time and are delicate it is hard to know in what sense the errors caused by them may accumulate. In this case the possible errors may arise from:

(a) The inaccurate determination of the magnetic north-south (in connection with the magnetic east-west); this is the greatest and the most important error which may be made as it will probably affect the observations in most cases.

(b) The readings taken from the dial by means of the vernier. The taking of the readings should be well practiced, as this is not as easy as is usually expected. Even the most trained persons can make errors. The readings must be verified at least twice.

(c) The involuntary shocks produced by turning the support of the magnets K about its axis.

5. Temperature Constants

The determination of the temperature constant is the most delicate operation and requires long time. Let us suppose that the temperature coefficient of a magnetometer is to be determined. The sole mean for examination is to have two identical models, the temperature coefficient of one of which is exactly known and which serves simply for the determination of the diurnal variation. Under these conditions the latter can be very well replaced by a corresponding magnetograph (recording apparatus). The method adopted at the "École Nationale Supérieure du Pétrole" is the following: The magnetometer under examination is put in a small, well-isolated room (to avoid the draught caused by heating); the temperature is gradually increased from the center (with a constant thermal gradient). The corresponding readings are made from the scale, taking care that a uniform temperature inside of the instrument is maintained. For this purpose an endeavor is made to obtain an accord of about one-tenth of 1° between the readings of the two thermometers, the one in the room and the other in the apparatus; a magnifying glass may be used for reading if necessary. During several days (about ten) readings are made twice a day, in the morning and in the evening. As the corresponding diurnal variations are known exactly, the mean value and, as accurately as possible (at about 1/10), also the temperature coefficient can be deduced.

To make the error as small as possible it is necessary to use always the method of the least squares. The value obtained in this way is usually very correct. But there arises the following question. What may happen if the constant thermic gradient is no longer maintained? During our own investigations we established that the temperature coefficient μ does not have a constant value but is to a certain degree a function of the thermic gradient; of course, this difference does not, in general, much affect the precision of the final results.

In case of Haalck's magnetometer (No. 36, constructed by Exploration) the measurements were carried out, according to the method indicated above, at a mean temperature of 22° C; for the vertical component the variation of μ was between 7.74 (for a variation of 0.2° C. per minute) and 7.96 (for a variation of 1° C. per minute). For practical use we adopted μ = 8.0; owing to the fact that the readings of the thermometer could be easily made by the naked eye with an accuracy equal to about ± 2/10 of a division, the absolute error did not exceed 3 gammas; this error can be considered acceptable.

Thus we are of the opinion that, in order to reduce the probable errors, instruments with the smallest possible temperature coefficients should be acquired. These values in balances manufactured by Askania are in general from three to five.

The method of heating can be accomplished satisfactorily by using an electrical radiator regulated by inserting a rheostat in the circuit.

Some authors prefer to put the magnetometer into a box, the temperature of which is gradually increased by electrical heating. This method seems to be very satisfactory; some others work simply in the open air, but this practice must be rejected, as there is never a uniform thermal stability between the instrument, the coefficient of temperature of which is to be established, and the ambient air. Even under the best conditions there will always be a difference between the temperature indicated by the thermometers of the instrument and the real temperature.

During the beginning and the end of different operations made in the field at one station the temperature varies often from 3 to 4° C., especially if the work is done under full sunshine at about 11 a.m.; as the variations of the thermal conditions of the magnetic system and the indications of the thermometers are not simultaneous, there is necessarily a difference between them.

The temperature shown by the thermometer at a certain moment of operations may thus not correspond to the exact correction to be applied to the results. The thermometer indicates the same temperature as the magnetic system at a moment when the temperature of the latter is higher (the temperature supposed to be increasing). Thus it is desirable to know this difference and in particular the inertia of the thermometers of the instrument. Accordingly, always taking the mean value of the initial and final temperatures; and often even to take into account, in case the thermic gradient is very high, a third reading made between the observations.

6. Variation of the Horizontality of the Apparatus

In the first place it must be verified whether the level of the instrument is horizontal with respect to its support; if not, it must be regulated according to the rules applied to topographic instruments. Besides, at

every station it must be examined whether the level has been deranged or not by involuntary shocks caused during transportation. As a rule the horizontality of the instrument must always be regulated carefully and it must be observed during the whole time of operations that the bubble is not displaced at more than one-half division. On the other hand, if the soil is wet and soft, the tripod may be deranged without being noticed, and it results that the large readings made from the scale may be attributed involuntarily to the local anomalies, although in reality they are due only to these displacements of the tripod. The same phenomenon may occur in case of a frozen ground.

In putting the instrument at a station it is necessary to drive the tripod legs well into the ground and to regulate the table so as to make it almost horizontal before the apparatus is placed on it. By this the operating of the three levelling screws up to the end of their windings is avoided.

7. Condition of the Knife Edge

The knife edge is the most delicate part of the instrument, as the whole sensitiveness depends on it. Greatest care must be taken by the operator to keep it in good condition. Therefore it must never be forgotten to clamp the magnetic system after the completion of operations at a station. The knife is of agate or quartz, which are very fragile; all shocks of the edge against the pivots, which are of the same material, should be avoided. All these accessories are perfectly elaborated; the edge may be of cylindrical, circular, or parabolic form.

From this fact it is clear a priori that the constant of the scale can not have the same value at the different parts of the scale, because the distance of the center of gravity of the magnetic system to its axis of oscillation varies when inclined.

In order to find the relation by which the coefficient of the scale is connected with the inclination of the magnetic system of the variometer, the method applied in the laboratories of the École Nationale Supérieure du Pétrole et des Combustibles Liquides is the following:

The magnetic system is given increasing inclinations by intermediation of a bar magnet P placed under the instrument over a graduated ruler (second position of Gauss); after this the relative position of this bar with regard to the center of oscillation of the magnetic system is changed, and at each position of it, which must be well determined, the constant of the scale is determined by the frame method.

After this the curve K = function E is traced, in which K is the constant of the scale and E the inclination expressed in divisions of the scale.

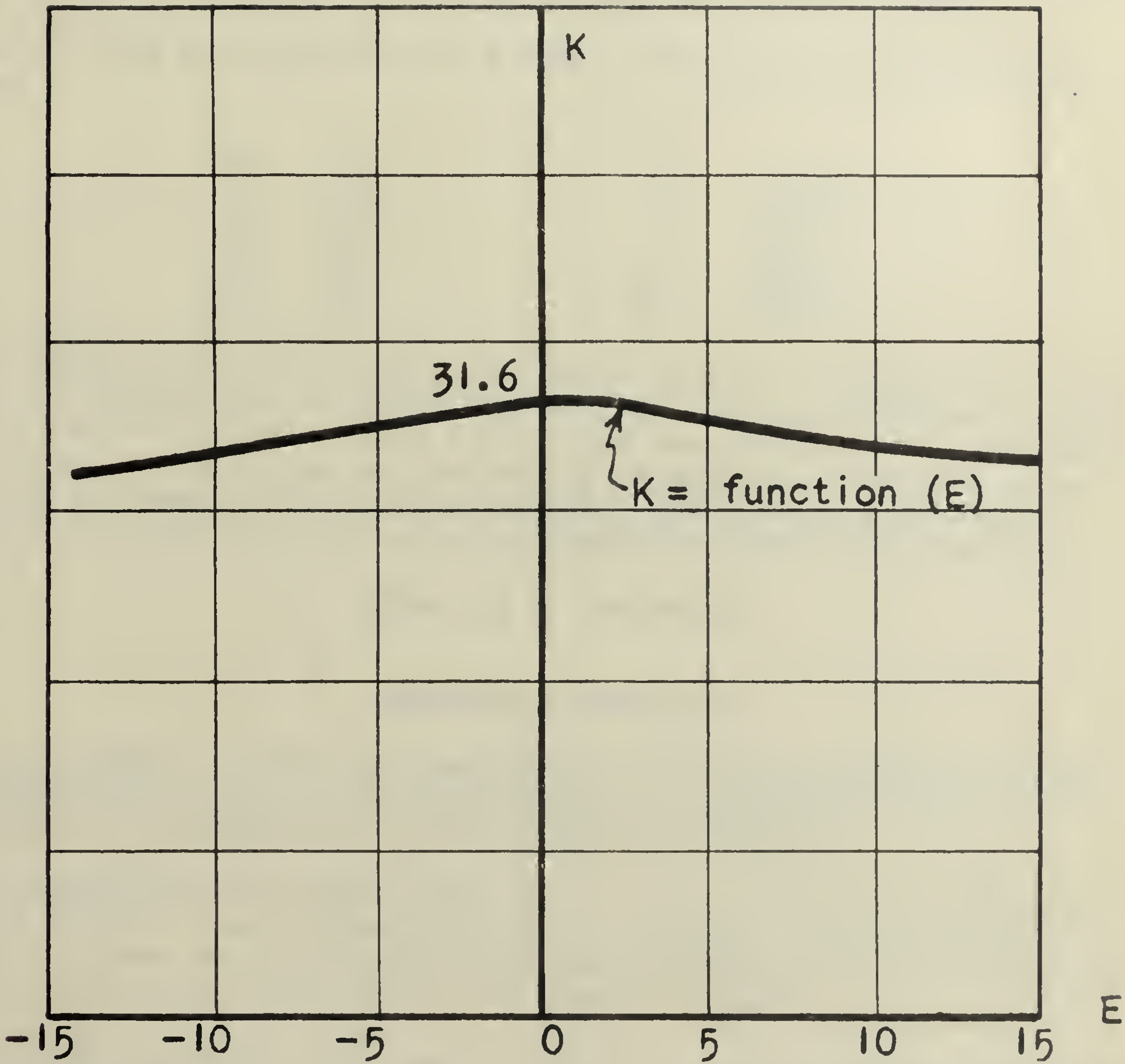

Figure 1.- Variation of scale constant K with inclination E, for Haalck's variometer, method of the École Nationale Supérieure et des Combustibles Liquides

Thus by application of this method to the vertical component of Haalck's variometer we obtained the curve shown in Figure 1.

This curve is slightly dissymmetrical with regard to the axis of ordinates and it shows that K diminishes in case of strong inclinations of the magnetic system. This means that the sensitiveness of the apparatus increases with the increase of inclination. These results are in full accord with those obtained by the United States Coast and Geodetic Survey.

In our case the mean values of K which we deduced from tests are as follows:

E	K	E	K
0	31.6	0	31.6
+ 5	31.4	- 5	31.2
+10	31.0	-10	30.8
+15	30.6	-15	30.4

Contrary to the usual opinion, the knife edge is not spoiled very quickly. we can even declare that it can be used as long as the magnetometer, if the magnetic system is liberated with special care and always in the same manner. Rough turning of the handle must be avoided especially. If a harsh noise is heard from the shock of the knife against the pivot, this is unfavorable. Locking the instrument before being transported must never be forgotten.

II. ERRORS DUE TO OBSERVATION

1. Selection of Stations

In selecting a station for measurements the following should be taken into consideration: (a) its accessibility, (b) the physical conditions of the ground, (c) and the direct action of the sun and wind.

(a) Stationing on a sloping place or on a road, unless the latter remains for the most time unfrequented, must be avoided. In case of the necessity to reach the station across an uneven terrain the apparatus should be carried with great precaution by the operator himself.

(b) Lawns and soft ground should be avoided. Before starting the measurements it is necessary to examine whether the movements of the operator around the tripod affect the horizontal position of the instrument or not. The operator must be sure that the instrument is established firmly.

(c) It must be tried to remain always in the shadow, at the same time avoiding the draughts. A place between two stems of large trees where the draught is sometimes very great should be avoided; in this case the error may be caused not only by the variation of the thermal conditions around the

instrument, but also by the mechanical action of the wind by which the instrument can be moved in an irregular way. In selecting the base stations the operator must be careful to see that these stations are located so as to remain in the shadow during the whole day. If the frequent returning during the day to the base station is not convenient, it is preferable to select several base stations and then unite them into one station.

Finally, a station must always be marked by a number or a letter and the date of the operation should not be forgotten..

2. Setting the Instrument

All the objects which can affect the magnets should be left behind when going to the field. It is a bad habit to remove the objects at each station and then to carry them after the completion of the operation to the next station. Special care must be taken that such objects as, for example, wrist watches, keys, iron belt buckles, purses, iron and nickel plated cigarette holders, chains, necktie pins, buttons, and nailed boots are not used. The operator must always make sure that there are no magnetic objects around the instrument.

Telegraphic posts, electrical lines and towers, wires of barbed iron, the proximity of villages and especially of factories must be avoided as much as possible. Stations sites must be selected at least 10 kilometers from large cities.

In placing the instrument upon the table of the tripod the hooks must be closed without using too much force.

3. Position of the Operator

In making the readings the operator must try to occupy always the same position with regard to the magnetometer. It must always be remembered that the apparatus is very delicate and expensive. By doing so the operator will instinctively pay great attention to it during the operations. The fundamental principle that an apparatus should be used by the same operator only, and must never be loaned, should also be adopted. Nobody should be permitted to approach the apparatus; even the assistant must stay away at about 10 meters distance. During the readings the operator should cross the hands behind his back; he must try to make the readings by opening both eyes equally in order to avoid the parallax error and besides in order not to impair the sight. Before starting the work on a series of stations the operator must not forget to adjust the telescope.

The opening of the box of the apparatus in order to explain its function to the visitors must be avoided; each operation of such kind injures the sensitiveness of the instrument.

4. Carrying Out of Measurements

Two persons are sufficient for carrying out measurements with one apparatus: The operator, who takes the readings and transports the apparatus, and his assistant, who writes down the results and makes the necessary numerical calculations; the latter measures also the distances between the respective stations and transports the accessories (boxes, maps) if any. The selection of the stations is made by the operator who chooses the places considered to be the best for the measurements. The assistant must know how to find the correct places of these stations on the map. Some operators prefer to hire assistants from the population of the region in which they are working. This can not be recommended as the quality of the work becomes very poor. In the first place the assistant must know what he has to do, as it is very disagreeable to explain to him every step of his new occupation. Calculations required on the spot during the time the operator adjusts his instrument will besides be certainly above his intellectual faculties. Finally, and this is important, he can not give explanation if during the day the operator or even he himself has made a mistake or has forgotten to carry out some operations. Therefore it is necessary that the assistant also be, to a certain degree, able to give advice to the operator. Finally, after each station, the operator must make certain that the magnetic system of the instrument is well balanced and that the safety device is well inserted.

5. Reading From the Scale

Before taking a reading the scale must be perfectly lighted in order to avoid the formation of the image of the sun on the scale. Besides, if the lighting is very bright the use of the rotating mirror becomes unnecessary.

During the reading the operator must be able to see clearly at least 1/10 of one division of the scale; this corresponds to the measurements equal from 2 to 3 gammas (relative error); some operators are even able to read about 1/20 of a division. This is usual with our pupils who are trained in geophysical prospecting.

After having made a reading the operator must ask the assistant to repeat it accurately. The same brief and precise expressions should be used always; this will make the work methodical and even mechanical.

The readings of the temperature before and after the operation should not be forgotten. The assistant must watch all the operations of the operator Finally, after the termination of readings at a station the operator must ask himself whether he did not forget something.

6. Determination of the Magnetic East-West. Influence of the Bad Orientation of the Instrument

(a) If Lloyd-Schmidt balance is used it is necessary that in putting the compass on the table of the tripod the determination of the east-west magnetic direction is made with an accuracy equal to at least $\frac{1}{2}^{\circ}$; the graduation of the compass is divided into degrees. After a little training readings with an accuracy equal to $1/3^{\circ}$ can be made.

(b) If Haalck's variometer is used, the determination is much more delicate; readings from the limb can be made with an accuracy equal to 30" of the arc. An orror in orientation equal to 1' of the arc produces easily an error equal to 3 gammas for an instrument the scale constant of which is of the order 30 (for the two components of the terrestrial magnetic field). In general the sensitiveness of Haalck's variometer is higher than that of Lloyd-Schmidt balances; this can be proved by using both instruments in the field, as well as from the theory of their construction.

Finally, by using Haalck's variometer attention must always be paid to a perfect determination of the east-west magnetic direction of the base station because the accuracy of all the further results depends on it.

7. Final Calculations

As a rule, final calculations are made in the evening after the conclusion of the work in the field. We recommend, of course, that preliminary calculations, giving rough results, be made at every station. In this case if a considerable anomaly is established the operator must immediately return to the previous station and find out whether this anomaly is not due to the shift of zero point of the scale between the two stations; if the same value as obtained before is found, the operator can be assured that the measurements are correct.

Final formulas always give variations of the two components of local magnetic field in gammas ($1\gamma = 10^{-5}$ gauss).

These formulas contain:

1. Readings made from the scale.
2. Exact values of the constants of the scale.
3. Diurnal variations.
4. Regional corrections (longitude, latitude).
5. Temperature constants.

Values of magnetic moments of the auxiliary magnets, as well as those of the magnetic system of the instrument and the regional values of the two components of the terrestrial magnetic field, enter indirectly into the expression for the constants of the sacle. Notwithstanding the variety of these different quantities the application of these formulas is easy and entirely mechanical.

III. ERRORS DUE TO NATURAL CAUSES

1. Displacement with Regard to Geographic Coordinates

In moving along a wide region the terrestrial magnetic field is submitted to a continuous variation in longitude and in latitude without the intervention of any disturbing foreign magnetic body. Thus, on the terrestrial globe, the vertical component varies from -0.67 to +0.63 Gauss (counting from the south pole toward the north magnetic pole), while the horizontal component, considering it positive on the whole surface of the earth, varies from zero (at the magnetic poles) to 0.39 Gauss (equator). Under these conditions the corrections to be applied are about, for the vertical component:

In latitude, 8 gammas per kilometer,
In longitude, 1 to 2 gammas per kilometer.

The latter correction can usually be neglected in practice; it may be taken into account if the anomalous curves are directed from east to west.

For the horizontal component:

In latitude, 5 gammas per kilometer,
In longitude the correction is negligible.

It must not be forgotten that if the movement proceeds toward the north the corrections are:

To be subtracted for the vertical component;
To be added for the horizontal component.

If the movement proceeds toward the south, the operations are to be effectuated in the inverse sense.

Thus the magnetometer used is adjusted for a relatively limited region. As a practical rule it can be adopted that this area is extended toward the north and the south from the base station which was used for the adjustment of the constant of the scale of the magnetometer. Beyond these limits it is necessary to make adjustments for the new region. The use of a magnetometer must, approximately, be limited by an arc equal to 220 kilometers of a terrestrial meridian, say 2° of difference in latitude, for example, of the length of the Rhine Valley (Fossé rhénan).

2. Diurnal Variations

It is known that the terrestrial magnetic field displays secular, annual, and diurnal variations. The two first are not important to the prospector, due to their extreme slowness. The effect of the diurnal variation is on the contrary very intense during certain hours of the day; therefore we recommend working in the fieldwithout interruption from 10 a.m. to 3 p.m. This is the time of the day when the diurnal variation of the terrestrial magnetic field is the smallest.

The results of the measurements of the diurnal variation may be corrected in the following ways:

(1) By taking the exact values of this variation from a magnetic observatory--for example, for France and Algiers from the observatory of Vol-Joyeux.

(2) By installation of a temporary observatory in the center of the region to be prospected by mounting there a magnetometer and assigning a special observer. This method is the best because the diurnal variation of the terrestrial magnetic field is not entirely identical in all the parts of the same country. This method requires, of course, additional expense, thus it can be used only by comparatively rich organizations. Its great advantage consists of the fact that the different values of the diurnal variation can be obtained immediately. In this case the work can be carried on from 9 a.m. to 5 p.m.

(3) For organizations which are less fortunate the following manner of work can be recommended:

Mean monthly values for each hour of a day of diurnal variations of the two components of the terrestrial magnetic field of the previous year or of the year the results of which were published by the Institut de Physique du Globe de l'Université de Paris are taken and curves showing these variations are traced at a suitable scale; this will give 12 graphs which can be used during the operations in the field.

Of course values with an approximation of from 3 to 4 gammas only can be obtained, but this error is acceptable. According to experience obtained in the field it can be stated that this method gives very satisfactory results from the viewpoint of industry.

(4) The last procedure is that of making the work a mechanical one by using for the two components recording instruments called magnetographs. The results obtained by these instruments are, of course, not as accurate as desirable. Their sensitiveness is not very great and they are affected, especially by the variation of the temperature. The registration is made in these instruments photographically, but its enlargement is not sufficient, the course of the luminous ray being very short. Besides, they are too heavy for convenient transportation, their handling requiring great care, suitable installation, and trained personnel. Thus, taking into consideration the great expenses and difficulties caused by their transportation it is better not to use them.

As a rule the amplitude of the diurnal variation of the terrestrial magnetic field is twice as great in summer as in winter. With regard to the months the courses of the curves of the diurnal variation are the same for the following series of months:

January, February, November, and December.
March, April, September, and October.
May, June, July, and August.

All the curves show a minimum at noon. There is a perfect concordance of the same series of the months of the year with the variation of the mean temperature. In winter the variation of the temperature is very small, owing to the fact that cloudy days are the best for the work of the prospector (the most uniform temperature); the winter is theoretically favorable for the magnetometer work, but unfortunately during this season the atmosphere conditions make the work often impossible.

3. Magnetic Storms

There are often days (especially during the summer) when the magnetic measurements seem to be entirely incoherent. This is due to the irregular variations of the terrestrial magnetic field, called magnetic storms.

In case of such storms it is absolutely useless to continue the measurements, even if the storm lasts several days. In order to know in advance the date of the storm it is recommended to stay in close relation with a magnetic observatory from which the information on the storm may be obtained by wire. If the operator works far away from civilized places the presence of a magnetic storm can be established by him from the irregular courses of the profiles. A person sufficiently trained in magnetic survey may easily establish this.

4. Electrical Currents

Networks of direct current are fatal to the magnetic survey, thus they must always be avoided. The effect of the alternating current, with which the surveyor has usually more chance to come into contact, can, on the contrary, be almost neglected; in this case the sole inconvenience arises from metallic poles; 150 to 200 meters distance should be kept between the poles and the instrument. Electric traction often produces considerable effect on the measurements.

In order to establish this it is sufficient to put a magnetometer at 100 meters distance from a moving electric vehicle (tramway or railway); the parasitic electromagnetic effect attains a value from 80 to 100 gammas.

IV. CRITICISM OF THE RESULTS OF MEASUREMENT

1. Accuracy of Measurements

(a) The total error which can be allowed to be made by a trained and conscientious operator should not exceed ± 10 gammas. This error can be apportioned as follows:

From 2 to 3 gammas to the reading from the scale;
From 2 to 3 gammas to the reading of the thermometers;
From 3 to 4 gammas to the estimation of the diurnal variation.

(b) In order to estimate the accuracy of the measurements it is recommended to make, as often as possible, the measurements twice at the same station and at different intervals. Thus, let us suppose that in order to determine the values, the variation of the vertical component is measured and

$$\Delta Z_1;\ \Delta Z_2;\ \ldots\ldots\ldots;\ \Delta Z_n.$$
$$\Delta Z'_1;\ \Delta Z'_2;\ \ldots\ldots;\ \Delta Z'_n.$$

are the two series of the measurements made at stations 1, 2 ... n. The differences

$$\begin{aligned} \Delta Z_1 - \Delta Z'_1 &= \varepsilon_1 \\ \Delta Z_2 - \Delta Z'_2 &= \varepsilon_2 \\ \hline \Delta Z_n - \Delta Z'_n &= \varepsilon_n \end{aligned}$$

represent the relative errors made at these stations; the mean quadratic error will be:

$$\varepsilon = \pm\frac{1}{n}.\sqrt{\varepsilon_1^2 + \varepsilon_2^2 + \ldots + \varepsilon_n^2}.$$

The operator must obtain from his measurements $\varepsilon \leq 10$ gammas. Therefore the consideration of the mean quadratic error is, for a quick estimation of the measurements obtained, very useful.

3. Coordination of Stations of Measurement

Very often the geologist is especially interested in obtaining for his investigations the absolute value of the terrestrial magnetic field. This determination by means of absolute instruments is tedious and difficult, especially if a great area is to be surveyed. In this case the difficulty can be removed by attaching to the absolute instruments a magnetic variometer, The technique of the work will then be as follows:

A limited number of stations of absolute measurements in the region under consideration is chosen and then the stations relative to the first ones are coordinated by means of variometer measurements. Owing to the fact that the sensitiveness of absolute instruments is of the same order (if not higher) as that of the present variometers, the coordination can be made without difficulty.

Of course, care must be taken as to the manner in which the coordination is carried out, as this coordination can be done in many ways; the simplest method giving the highest possible accuracy must be selected.

In using the magnetometer, the application of the method of least-square adjustment can not be recommended, as it requires the use of the calculus. It is better to depend on accurate instruments and good operators than to have the illusion of possessing faultless results based on abstract mathematical theories, because the latter are justifiable only if the measurements made in the field are correct; but in this case there is no necessity for these theories.

The surveyor must try to be a good operator in the field and an ardent calculator in the office. But he must take care not to use his mathematical knowledge beyond the reasonable and necessary limits.

3. Magnetic Profiles

Two kinds of profiles may be used, according to necessity:

(a) Closed profiles.
(b) Open profiles.

Linear profiles come into the second category.

(a) The case of a closed profile does not occur usually in the magnetic prospection; it may be carried out in the case of localizing the borders of a mineral deposit; the error of the closure is then distributed linearly along the whole profile. Often, if there is only one absolute station and the surveyor wants to adjust to it the relative stations, it is preferable to distribute the latter around it and to carry out a certain kind of closed profile.

(b) The most usual case is that of open profiles; particularly that of rectangular profiles. This is the most used representation of magnetic prospection. On the same sheet of paper divided into squares or on a board showing the distribution of stations the consecutive profiles can be represented in the same way as the geological cuts (method used by Swiss school); in this case the interpretation is the easiest; the anomalies, as well as the various values at each station, are located on the map automatically.

In tracing the magnetic profiles the rectilineal system of coordinates is used; on the abscissa are drawn the mutual distances of the stations of measurement and on the ordinate the values of the anomalies measured, both of them at a convenient scale. It is recommended not to choose exorbitant scales. If there is no anomaly the anomaly which the surveyor may want to see will not appear by increasing the scale of the graphs. The scale to be chosen

depends necessarily on the order of magnitude of the anomalies; thus its choice depends solely on the good sense of the operator.

4. Reconnaissance Profiles. Magnetic Charts

Before starting the execution of the network of the magnetic stations it is preferable to first make two magnetic profiles called the reconnaissance profiles. It is preferable to make them in the directions NE.-SW. and NW.-Se.; their intersection can be taken as a base station common to both of them. These reconnaissance profiles may have also the directions N.-S. and E.-W.; but according to the experience the first direction must be considered more suitable. If the time is limited or one reconnaissance profile seems to be sufficient, it is preferable to carry it out in the N.-S. direction for the sole reason that in general the geological formations are magnetically polarized in this direction and therefore there is maximum possibility to disclose the characteristic anomalies.

As soon as the existence of an anomaly is established it is necessary to cover the region with a network of stations, placed at regular distances. The most efficient method is the following:

The stations are distributed in a regular manner on two series of equidistant profiles intersecting one another at a right angle (preferably), so that the whole region to be explored is covered by a network of squares, the summits of which are occupied by the stations of measurement.

The mutual distances of the stations and the profiles must be between 50 and 150 meters, according to necessity; from the viewpoint of magnetic prospection distances of more than 150 meters must be considered too great.

In the course of exploration, measurement stations where the anomalies were found must be placed closer together than in other territory.

5. Representation of Numerical Operations

A constant system of calculations carried out by the assistant at the stations of measurement should be adopted. For the vertical component (the case when the Lloyd-Schmidt balances are used) the following system has been adopted:

N_w (Reading towards the West)		N_e (Reading towards the East)	
....		....	
....		....	
....		....	
....	(mean value)	....	(mean value)

$$\frac{n + n}{2} = \ldots\ldots\ldots$$

Temperature	(Beginning of the measurement. ((End of the measurement.	Hour	(Beginning of the (measurement. (End of the measurement.

At the bottom of the sheet: Final calculations concerning the value of ΔZ.

This system can, of course, be varied by different operators. We recommend choosing the system which seems to be the most convenient. The chief point is to obtain a working notebook which may be easily deciphered.

6. Number of Stations Which can be Completed Per Day

The highest ambition of operators, especially if they are beginners, is to complete a great number of stations during one day.

But it must be kept in mind that 15 stations with reliable data are better than 40 with doubtful ones. It is recommended that not more than 6 magnetic stations should be completed in one hour; these 10 minutes per station (including the time necessary for traveling about 100 meters) are sufficient only for the best trained operators. As a general rule it can be considered that by working from 10 a.m. to 3 p.m., without interruptions, a maximum of 25 stations can be completed. Besides, toward the end of the day the fatigue and the enervation of the operator must also be taken into consideration, as these two factors may greatly reduce the accuracy of reading.

I. C. 6528 July, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXVII

BY

Frederick W. Lee

I.C. 6528
July, 1931

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 27

Compiled by Frederick W. Lee[2]

List of contributing editors of Geophysical Abstracts:
Ayvazoglou, W., U. S. Bureau of Mines, Department of Commerce, Washington, D. C.
Barton, Dr. D. C., Petroleum Building, Houston, Tex.
Belluigi, Dr. Arnaldo, Corso Vittorio Emanuele 178, Parma, Italy.
Bogoiavlensky, Prof. L., Central Chamber of Weights and Measures, Leningrad, U.S.S.R.
Eckhardt, Dr. E. A., 327 Craft Ave., Pittsburgh, Pa.
Eve, Dr. A. S., McGill University, Montreal, Canada.
Gish, Dr. O. H., Carnegie Institution, Broad Branch Road, Washington, D. C.
Gorsky, Eng. V., Allatini Mines, Ltd., Skoplie B.p. 134, Yugoslavia.
Hartley, Kenneth, 2404 San Jacinto St., Houston, Tex.
Hutchinson, Prof. W. Spencer, Mass. Institute of Technology, Cambridge, Mass.
Jenny, Dr. W. P., Magnolia Petroleum Co., Dallas, Tex.
Karcher, Dr. J. C., Dallas, Tex.
Keys, Dr. D. A., McGill University, Montreal, Canada.
Knappen, Dr. R. S., Gypsy Oil Co., Tulsa, Okla.
Korzujin, Prof. J., National University of Mexico, Mexico, D. F.
Lane, Prof. Alfred C., Tufts College, Boston, Mass.
Lee, Dr. F. W., U. S. Bureau of Mines, Department of Commerce, Washington, D. C.
Leonardon, E. C., 25 Broadway, New York City.
Numerov, Prof. Dr. B. V., Fontanka 34, Leningrad, U. S. S. R.
Petrowsky, A., Wasilly Ostrov, 21 Linia No. 8-A, Leningrad, U.S.S.R.
Roman, Dr. I., 90 Valley Way, West Orange, N. J.
Ruark, Dr. A. E., University of Pittsburgh, Pittsburgh, Pa.
Scholl, Louis A., Box 1805, Houston, Tex.
Shaw, Dr. H., The Science Museum, South Kensington, London, S.W. 7.
Sundberg, Dr. Karl, Swedish American Prospecting Corp., 26 Beaver St., New York City.
Truemann, O. H., Humble Oil Co., Houston, Tex.
Van Orstrand, Dr. C. E., Interior Building, Washington, D. C.
von Weelden, Dr. A., De Bataafsche Petr.Mij.30 Carel van Bylandtlaan. The Hague, Holland.
Weaver, Paul, Drawer C, Houston, Tex.
Wright, Dr. F. E., Carnegie Institution, Washington, D. C.
Zuschlag, Dr. Theodor, Swedish American Prospecting Corp., 26 Beaver St., New York City

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6528."
2 - Senior physicist, U. S. Bureau of Mines.

This information circular contains abstracts by W. Ayvazoglou from original German patents concerning electrical methods of geophysical prospecting only. All patents issued since 1904 are considered.

Abstracts from German patents concerning other methods of geophysical prospecting will be published in special issues of Geophysical Abstracts appearing later.

Copies of the patents listed in this circular may be procured from Deutsches Reich, Reichspatentamt, Berlin, Germany.

4. ELECTRICAL METHODS

(240) VERFAHREN ZUR AUFFINDUNG UND BESTIMMUNG VON ERZLAGERN

(METHOD FOR DISCOVERING AND DETERMINING ORE DEPOSITS)

The Electrical Ore Finding Co., Ltd.,
of London

152,519 Patent issued July 5, 1904.

This invention relates to a method for discovering ore and other mineral deposits. It is characterized by the possibility of determining the property and the position of the deposit by means of sound produced in a telephone receiver inserted between two portable electrodes and by which a contact with the surface of the earth is established, while the variable electric current impulses are sent through the ground from an induction coil by means of another pair of portable electrodes.

The whole arrangement consists of (1) an induction coil, (2) a telephone-receiver, and (3) four electrodes for the establishment of earth connections.

Claims allowed - 1.

(241) VERFAHREN ZUR SYSTEMATISCHEN ERFORSCHUNG DES ERDINNERN GRÖSSERER GEBIETE MITTELS ELEKTRISCHER WELLEN

(METHOD OF SYSTEMATIC EXPLORATION OF THE SUBSOIL OVER LARGE AREAS BY MEANS OF ELECTRICAL WAVES)

Dr. Gotthelf Leimbach and Dr. Heinrich Löwy
of Göttingen

237,944 Patent issued Sept. 14, 1911.

The patent discloses a method for locating hidden ore deposits by the erection on the area to be explored (level country, mountains, islands) of antennae by which the electrical waves are emitted, or received. On a level terrain about 200 meters deep holes are drilled in which the antennae

wires are inserted. In mountainous regions the antennae are erected at the foot of mountains. In case of the exploration of islands the antennae wires are sunk in the water from ships.

Claims allowed - 1.

(242) VERFAHREN ZUM NACHWEIS UNTERIRDISCHER ERZLAGER ODER VON GRUNDWASSER MITTELS ELEKTRISCHER WELLEN

(METHOD FOR DETERMINING SUBTERRANEAN ORE DEPOSITS OR UNDERGROUND WATER BY MEANS OF ELECTRICAL WAVES)

Dr. Heinrich Löwy and Dr. Gotthelf Leimbach of Göttingen)

246,836 Patent issued May 11, 1912.

The method described in this patent consists of an arrangement by which electrical waves are emitted from the antenna of a sending system established on the surface of the ground. After the reflexion of these waves from the subterranean body the latter is disclosed by the receiving station, also provided with an antenna.

Claims allowed - 1.

(243) VERFAHREN ZUR ERFORSCHUNG VON GESTEINSSCHICHTEN INNERHALB VON BERGWERKEN

(METHOD OF EXPLORATION OF ROCK-LAYERS WITHIN MINES)

Dr. Heinrich Löwy of Göttingen

254,478 Patent issued Dec. 6, 1912.

This patent discloses a method for exploration of rock-layers within the mines by producing inside of drifts and boreholes electrical resonance currents and measuring there their capacity (frequency) and damping.

Claims allowed - 1.

(244) VERFAHREN ZUM NACHWEIS UNTERIRDISCHER ERZLAGER UND GRUNDWASSERSPIEGEL MITTELS ELEKTRISCHER WELLEN

(METHOD FOR DETERMINING SUBTERRANEAN ORE DEPOSITS AND THE LEVEL OF UNDERGROUND WATER BY MEANS OF ELECTRICAL WAVES)

Dr. Heinrich Löwy of Göttingen

254,517 Patent issued Dec. 6, 1912.

According to this patent, subterranean ore deposits and the level of underground water may be disclosed by observation of the interference, in a

receiving antenna stretched horizontally close above the ground, of electrical waves which are produced in a sending antenna running parallel to the receiving antenna, the waves being received partly directly and partly after their reflection from the subterranean ore deposit (or from the level of the ground water).

Claims allowed - 1.

(245) VERFAHREN ZUR BESTIMMUNG DER BESCHAFFENHEIT DES ERDBODENS MITTELS ELEKTRIZITÄT

(METHOD FOR DETERMINING THE CHARACTER OF THE SOIL BY MEANS OF ELECTRICITY)

Conrad Schlumberger of Paris

269,928 Patent issued Feb. 5, 1914.

This patent discloses a method for determining the character of the soil by the following series of steps: (1) Obtaining around two points (electrodes) on the surface of the earth, of equipotential curves and comparing them with equipotential curves determined theoretically for homogeneous ground; (2) determining equipotential curves produced by polarization at the borders of an ore deposit; (3) preventing disturbances caused by earth currents, polarization of electrodes, etc., by periodical commutation of the current flowing between the electrodes; (4) determining the potentials of the ground by means of a movable line in which a galvanometer is included, the line being provided at both ends with metallic rods inserted into a porous receptable filled with a concentrated solution of copper sulphate or a similar salt, in order to establish a nonpolarizing earth-connection; (5) determining the relative potential drop between the equipotential curves with respect to the potential drop between two fixed points on the ground by means of a suitable method of measurement (Wheatstone bridge).

Claims allowed - 5.

(246) VERFAHREN ZUR BESTIMMUNG DER BESCHAFFENHEIT DES ERDBODENS MITTELS ELEKTRIZITÄT

(METHOD FOR DETERMINING THE CHARACTER OF THE SOIL BY MEANS OF ELECTRICITY

Conrad Schlumberger of Paris

272,603 Patent issued April 6, 1914.

This is an addition to patent 269,928 concerning the same subject. The method is characterized by determining equipotential curves which result from polarization currents produced by ore deposits and metallic masses hidden in moist ground, without being influenced by artificial sources of current.

Claims allowed - 1.

(247) VERFAHREN ZUR AUFSUCHUNG LEITENDER FLÄCHEN (Z. B. WASSER UND ERZ) MITTELS ELEKTRISCHER WELLEN

(METHOD FOR EXPLORATION CONDUCTING SURFACES (FOR EXAMPLE WATER AND ORE) BY MEANS OF ELECTRIC WAVES)

Dr. Gotthelf Leimbach of Göttingen

273,339 Patent issued April 24, 1914.

The method described is characterized by determining that length of a wave by which a sender, provided with one or several antennae running approximately parallel to the conducting surface, shows minimum damping, in which case one-fourth of the length of the wave or an odd multiple of it is equal to the distance of the antennae from the conducting surface.

(248) VERFAHREN ZUR FESTSTELLUNG DES VERLAUFS VON ERDBOHRUNGEN

(METHOD FOR DETERMINING THE COURSE OF BOREHOLES)

Erforschung des Erdinnern G.m.b.H., Hannover
Geschäftsstelle Göttingen in Göttingen

287,610 Patent issued Sept. 28, 1915.

The method for determining the course of boreholes, described in this patent, is characterized by an arrangement in which the metallic parts which may be found in the boreholes, especially the metallic tubing of, for example, two adjacent boreholes, are connected to an alternating current generator, so that they form the coating of a condenser the dielectric of which represents the intermediate ground. By measuring and determining the capacity, resistance, and self-induction the distance between the coatings of the condensers, that is, the distance between the two boreholes, is determined.

Claims allowed - 1.

(249) VERFAHREN ZUR ERFORSCHUNG VON GESTEINSCHICHTEN DURCH ELEKTRISCHE MESSUNGEN

(METHOD FOR EXPLORATION OF ROCK LAYERS BY MEANS OF ELECTRICAL MEASUREMENTS)

Erforschung des Erdinnern G.m.b.H., Hannover
Geschäftsstelle Göttingen in Göttingen

287,611 Patent issued Sept. 25, 1915.

This patent disclosed a method for electrical measurements by which the resistance of rock layers is determined with the aid of electrodes, buried in the rock, applying alternating current of suitable frequency

Claims allowed - 1.

(250) VERFAHREN ZUR AUFFINDUNG UNTERIRDISCH ODER VERDECKT VERLEGTER METALLISCHER KABEL ODER ROHRSTRÄNGE MIT HILFE SCHNELLER ELECTRISCHER SCHWINGUNGEN

(METHOD FOR DISCOVERING METALLIC CABLES OR PIPE-LINES BURIED IN THE GROUND OR PLACED UNDER SHELTER BY MEANS OF RAPID ELECTRICAL OSCILLATIONS)

Dr. Max Dieckmann of Gräfeling near Munich

303,912 Patent issued Sept. 29, 1919.

The method is characterized by the transmission, from a generating system to the conductor sought for, of high-frequency energy by means of coupling so that the high-frequency field produced around the conductor can be determined by means of an indicator of which the sensitivity of oscillations is either adjusted or not adjusted.

Claims allowed - 1.

(251) EINRICHTUNG ZUR AUFFINDUNG UNTERIRDISCHER ODER VERDECKT LIEGENDER, GUTLEITENDER GEGENSTÄNDE

(METHOD FOR FINDING SUBTERRANEAN OR SHELTERED OBJECTS OF GOOD CONDUCTIVITY)

Gesellschaft für drahtlose Telegraphie m.b.H. in Berlin

305,574 Patent issued Sept. 9, 1919.

This method is characterized by an oscillating system with a cathodic generator, an induction coil with a strong leakage field connected to it, and an arrangement for noticing the frequency changes of a resonance current which are caused when the induction coil is brought into proximity with the conducting body. Another feature of the invention consists of an arrangement for producing auxiliary oscillations which, combined with the original oscillations, result in an interference sound, so that the frequency change can be perceived either as a sound or as a change of the sound.

Claims allowed - 2.

(252) VERFAHREN UND VORRICHTUNG ZUM NACHWEIS UNTERIRDISCHER ERZLAGER ODER VON GRUNDWASSER MITTELS ELEKTRISCHER SCHWINGUNGEN

(METHOD AND ARRANGEMENT FOR DETERMINING SUBTERRANEAN ORE DEPOSITS OR UNDERGROUND WATER BY MEANS OF ELECTRICAL OSCILLATIONS)

Konstantin Schilowsky of Davos-Dorf, Switzerland

322,040 Patent issued June 19, 1920.

The invention discloses a method by which long waves are emitted over large subterranean areas to great distances in a horizontal direction,

as well as in depth, by means of a sending apparatus (for example an antenna sender); the wave energy produced by these waves, after having met an orebody and being scattered by it, is caught on the surface in the proximity of the deposit by means of portable receiving apparatus, the latter being protected from the effect of the primary field of the sending apparatus.

The receiving apparatus serving for catching the energy scattered by the orebody consists of closed resonance currents with large-diameter solenoids; these solenoids are protected from the effect of the primary waves emitted by the sender by arranging them so that their area of turn coincides with the direction of the magnetic vector of the primary waves.

Claims allowed - 2.

(253) EINRICHTUNG ZUR AUFFINDUNG UND FESTSTELLUNG VERBORGENER ODER UNZUGÄNGLICHER METALLE

(METHOD FOR DISCOVERING AND DETERMINING HIDDEN OR INACCESSIBLE METALS)

Dr. Max Reithoffer of Vienna

326,467 Patent issued Sept. 29, 1920.

The method considered in this patent is characterized by mutual connection of resonance coils, traversed by alternating currents of low and high frequency, in such a manner that the changes caused by introduction of metallic masses into the electrical or magnetic field of one or two of these coils of the value equal to the induced or the effective ohm's resistance (damping) or of the natural frequency of the coils concerned, result in potential differences which can be measured or observed by known methods.

Claims allowed - 5.

(254) VORRICHTUNG ZUM ENTDECKEN UND BESTIMMEN DER LAGE VON ERZ-ODER MINERALGÄNGEN AUF ELEKTRISCHEM WEGE

(METHOD FOR DISCOVERING AND DETERMINING THE POSITION OF ORE DEPOSITS AND MINERAL DEPOSITS BY ELECTRICAL METHODS)

Hans Torkel Fredrick Lundberg and Harry Johan Hjalmar Nathorst of Stockholm

328,835 Patent issued Nov. 8, 1920.

The arrangement disclosed in this patent consists of two pairs of electrodes; one of them is connected with the source of the current (primary electrodes) and the other (secondary electrodes, called searching rods) is provided with a sensitive receiver, for example, a telephone; the secondary electrodes are brought into connection with the ground at different places between the primary electrodes. The arrangement of the two primary electrodes (or only one of them) is proposed in the form of points lying along a straight

line or a curve--for example, in the form of a bare wire placed on the surface of the ground. It is very important that the extension occupied by the electrode (or electrodes) along the surface of the ground be great in comparison with the distance between the electrodes, as only in this case will the equipotential lines be straight lines or elongated curves.

Claims allowed - 1.

(255) VORRICHTUNG ZUM AUFFINDEN VON ELEKTRISCH LEITENDEN, DEM AUGE DURCH NICHTLEITER VERBORGENER KÖRPERN

(METHOD FOR DISCOVERING BODIES CONDUCTING ELECTRICITY, HIDDEN FROM SIGHT BY NONCONDUCTORS)

Ludwig Machts of Marburg, Lahn

330,090 Patent issued Dec. 7, 1920.

The patent discloses a method by which electric waves reflected by bodies conducting electricity are, after reflection, collected by a receiving apparatus. The method is characterized by an arrangement by which the reflected waves are united (similar to light waves) by means of a lens into a picture invisible to the eye; this picture is received automatically by a spiral antenna; the more or less strong induction currents produced in the latter are intensified by means of tubular receivers and intensifying lamps--for example, with the aid of a discharge line running synchronously to the antenna; the luminous discharges are projected upon a film and reproduced on it photographically; conclusions on the distribution of the conducting body can be drawn from the differences in the intensity of brightness of the films.

Claims allowed - 1.

(256) VERFAHREN ZUR FESTSTELLUNG VON ERZ-WASSER-ODER ANDEREN BODENSCHICHTEN MIT EINER VON DER UMGEBUNG ABWEICHENDEN ELEKTRISCHEN LEITFÄHIGKEIT MITTELS ELEKTRISCHER WELLEN

(METHOD FOR DETERMINING, BY MEANS OF ELECTRICAL WAVES, ORE LAYERS, WATER LEVELS AND OTHER SUBTERRANEAN LAYERS, THE ELECTRICAL CONDUCTIVITY OF WHICH IS DIFFERENT FROM THAT OF MATERIALS SURROUNDING THEM)

Eduard Raven of Gelsenkirchen

331,426 Patent issued Jan. 7, 1921.

The patent discloses a method in which the electric waves are emitted from two radio stations through the ground toward a common receiving station; from the sound picture received in the latter the nature and the extension of the disturbing body lying in the path of the waves can be determined.

Claims allowed - 1.

(257) VERFAHREN ZUR FESTSTELLUNG VON ERZ-WASSER-ODER ANDEREN BODENSCHICHTEN MIT EINER VON DER UMGEBUNG ABWEICHENDEN ELEKTRISCHEN LEITFÄHIGKEIT MITTELS ELEKTRISCHER WELLEN UND REFLEXION DERSELBEN

(METHOD FOR DETERMINING, BY MEANS OF ELECTRICAL WAVES AND REFLECTION, ORE LAYERS, WATER LEVELS AND OTHER SUBTERRANEAN LAYERS THE ELECTRICAL CONDUCTIVITY OF WHICH IS DIFFERENT FROM THAT OF MATERIALS SURROUNDING THEM)

Eduard Raven of Gelsenkirchen

331,427 Patent issued Jan. 7, 1921.

The patent discloses a method in which the electrical waves are emitted from two radio stations through the ground, and the paths of the waves, or their reflections, are determined by means of a receiving station situated at about an equal distance from the two sending stations and adjusted to the same sound.

Claims allowed - 1.

(258) VERFAHREN ZUR FESTSTELLUNG VON ERZ-WASSER-ODER ANDEREN BODENSCHICHTEN MIT EINER VON DER UMGEBUNG ABWEICHENDEN ELEKTRISCHEN LEITFÄHIGKEIT

(METHOD FOR DETERMINING ORE LAYERS, WATER LEVELS AND OTHER SUBTERRANEAN LAYERS THE ELECTRICAL CONDUCTIVITY OF WHICH IS DIFFERENT FROM THAT OF MATERIALS SURROUNDING THEM)

Eduard Raven of Gelsenkirchen

334,626 Patent issued March 15, 1921.

This patent discloses a method by which the disturbances of an electric current line field produced in the ground by layers of electrical conductivity different from that of the surrounding material, as well as the position of the disturbing mass, are determined by listening, on the surface of the earth or other accessible place, in a telephone inserted into a movable searching loop or coil. A device provided for amplifying the current--for example, a cathode tube--is inserted into the searching loop.

Claims allowed - 3.

(259) VERFAHREN ZUM AUFSUCHEN VON EISENERZLAGERN

(METHOD FOR DISCOVERING IRON-ORE DEPOSITS)

Eduard Raven of Gelsenkirchen

344,489 Patent issued Nov. 22, 1921.

The method for discovering iron-ore deposits described in this patent is based on the fact that the earthmagnetic lines of force existing at the

places of such deposits influence wire loops or coils (with or without an iron core) moved across the terrain; the weak induction currents thus produced are amplified by a suitable amplifier (for example a cathode-tube amplifier) so that they can be perceived by a telephone or a similar device. The coil is suspended inside of a frame which can be rotated around an axis perpendicular to the axis of the coil.

Claims allowed - 3.

(260) UNPOLARISIERBARE ELEKTRODE ZU ERDFORSCHUNGSZWECKEN

(NONPOLARIZABLE ELECTRODE FOR THE PURPOSES OF GROUND INVESTIGATION)

"Erda" Aktien-Gesellschaft of Göttingen

367,634 Patent issued Jan. 24, 1923.

This patent describes a nonpolarizable electrode used for ground investigation in which the metal electrode which serves as a means for passing of the current is surrounded by a series of solutions of similar or different metallic salts separated one from another by porous walls and having a concentration which approaches by steps the electrolytic concentration of the ground.

Claims allowed - 1.

(261) VERFAHREN ZUR ELEKTRISCHEN ERFORSCHUNG DES ERDINNERN

(METHOD OF ELECTRICAL INVESTIGATION OF THE INTERIOR OF THE EARTH)

Dr. Heinrich Löwy of Vienna

368,079 Patent issued Jan. 31, 1923.

The patent discloses a method of electrical investigation of the interior of the earth by which a large number of measurements can be made in a very short time by flying above the ground in an airplane or airship; the method is characterized by equipping the airship with an earth antenna drawn along the ground during the flight.

Claims allowed - 2.

(262) VORRICHTUNG ZUR AUFSUCHUNG UND LAGEBESTIMMUNG VON MASSEN ABWEICHENDER ELEKTRISCHER LEITFÄHIGKEIT IM ERDBODEN

(ARRANGEMENT FOR DISCOVERING AND DETERMINING THE POSITION OF SUBTERRANEAN MASSES HAVING ANOMALOUS ELECTRICAL CONDUCTIVITY)

"Erda" Gesellschaft für wissenschaftliche Erderforschung und Dr. Richard Ambronn of Göttingen

372,536 Patent issued March 29, 1923.

The arrangement for discovering and determining the position of subterranean masses having anomalous electrical conductivity is characterized by establishing the distribution of the electrical potential or the equipotential lines along the surface of the earth or along the boreholes by means of flat electrodes disposed on the surface of the earth or lowered into the ground in places suitably situated with respect to the object sought for.

Claims allowed - 1.

(263) VERFAHREN ZUR FESTSTELLUNG DER LAGE ELEKTRISCH LEITENDER FLÄCHEN IN DER ERDE MIT HILFE DER REFLEXION ELEKTRISCHER WELLEN

(METHOD FOR DETERMINING THE POSITION OF SUBTERRANEAN ELECTRICALLY CONDUCTIVE AREAS WITH THE AID OF THE REFLECTION OF ELECTRICAL WAVES)

Gesellschaft für drahtlose Telegraphie m.b.H. in Berlin

377,187 Patent issued June 13, 1923.

The patent discloses a method in which the sender consists of a horizontal antenna and the receiver of a frame antenna rotatable about a horizontal axis; by the latter antenna the angle of incidence of the reflected waves emitted from the conducting area, and therefore also the position of this area, are determined.

Claims allowed - 1.

(264) VERFAHREN ZUR FESSTELLUNG DER TIEFE EINER DAS STROMLINIENFELD STÖRENDEN EINLAGERUNG IN DER ERDE

(METHOD FOR DETERMINING THE DEPTH OF A SUBTERRANEAN DEPOSIT BY WHICH THE CURRENT-LINE FIELD IS DISTURBED)

Gesellschaft für praktische Geophysik, m.b.H. in Freiberg i. B.

377,837 Patent issued June 28, 1923.

The method disclosed in this patent is characterized by an arrangement in which two induction loops or induction coils assigned for the determination of the direction of the lines of force of the field are fixed at different heights from the surface of the ground and are rotated until maximum or

minimum induction effect is obtained. For the determination of the intensity difference of the field at the two positions the induction effects obtained in the two coils are compared one with another or determined, according to the zero method, with the aid of a branched part of the primary current working in the opposite direction.

Claims allowed - 1.

(265) VORRICHTUNG ZUR BESTIMMUNG VON RICHTUNG, INTENSITÄT UND POLARISATIONSZUSTAND ELEKTRISCHER WELLEN, INSBESONDERE ZU ERDERFORSCHERUNGSZWECKEN IN BERGWERKEN

(METHOD FOR DETERMINING THE DIRECTION, INTENSITY AND POLARIZATION STATE OF ELECTRICAL WAVES, ESPECIALLY FOR PURPOSES OF THE INVESTIGATION OF THE GROUND IN MINES)

"Erda" Aktien-Gesellschaft of Göttingen

378,224 Patent issued Nov. 16, 1923.

The method described in this patent is characterized by an arrangement in which the receiving system consists of a frame antenna rotatable about a horizontal and a vertical axis connected with a rotatable or oscillating condenser.

Claims allowed - 4.

(266) VERFAHREN ZUR AUFSUCHUNG VON KÖRPERN IM UNTERGRUNDE

(METHOD FOR DISCOVERING BODIES LYING UNDER THE GROUND)

"Erda" Aktein-Gesellschaft in Göttingen

392,158 Patent issued March 17, 1924.

This patent discloses a method for discovering bodies lying under the ground, the electrical conductivity of which is different from that of the material surrounding them, by means of electrical current-line method, equipotential-line method, or similar methods in case there is in the ground another known disturbing deposit the conductivity of which also differs from the material surrounding it. This method is characterized by screening off the known deposit from the field to be investigated by means of one or a series of auxiliary electrodes connected to one or both the field electrodes.

Claims allowed - 2.

(267) VERFAHREN ZUR ERGÄNZUNG VON MESSUNGEN MITTELS EQUIPOTENTIALLINIEN-METHODE FÜR ERDERFORSCHUNGSZWECKE ..

(METHOD FOR SUPPLEMENTING THE MEASUREMENTS MADE BY MEANS OF EQUIPOTENTIAL LINE METHOD FOR EARTH INVESTIGATION)

"Erda" Aktien-Gesellschaft in Göttingen

392,189 Patent issued March 17, 1924.

The method disclosed in this patent is characterized by an arrangement in which at places of measured potential lines, distributed suitably, the voltage drop perpendicular to the potential line is measured quantitatively or relatively and the local current density of the ground is determined in connection with the resistance measurements in the closing circuit of the probe current.

Claims allowed - 1.

(268) VORRICHTUNG ZUM AUFSUCHEN VON MAGNETISCHEN ODER ELEKTRISCH LEITFÄHIGEN STOFFEN

(METHOD FOR DISCOVERING MAGNETIC OR ELECTRICALLY CONDUCTIVE MATERIALS)

"Erda" Aktien-Gesselschaft of Göttingen

392,987 Patent issued March 28, 1924.

The patent describes an arrangement for discovering magnetic or electrically conductive materials based on their influence upon the development of lines of force in a magnetic direct current field or alternating current field or upon the induction coefficients of coils. The method is characterized by an arrangement in which two electromagnetic systems of different geometrical shape are in differential connection to each other; these systems are differently influenced by the distribution of the magnetically or electrically conductive masses in space.

Claims allowed - 3.

(269) ANORDNUNG ZUR BESTIMMUNG DER RICHTUNG EINES IN DER ERDE FLIESSENDEN WECHSEL-ODER GLEICHSTROMES

(ARRANGEMENT FOR DETERMINING THE DIRECTION OF AN ALTERNATING OR DIRECT CURRENT FLOWING INSIDE THE EARTH)

Gessellschaft für praktische Geophysik m.b.H. in Freiburg i. B.

393,644 Patent issued April 11, 1924.

This patent discloses an arrangement by which three search electrodes are connected to three or two coils or current conductors so that the angles of

the coil planes correspond to the angles formed by the lines connecting the three search electrodes; the effect of one coil corresponds to the length of the respective path on the earth, so that the direction of maximum or minimum induction in a rotatable coil or the deviation of a magnet needle shows the direction of the whole alternating or direct current inside the earth.

Claims allowed - 1.

(270) EINRICHTUNG ZUM NACHWEIS VON LEITENDEN KÖRPERN

(ARRANGEMENT FOR DETECTING CONDUCTIVE BODIES)

Rudolf Hans Richter of Leipzig-Gohlis

397,698 Patent issued June 28, 1924.

The investion discloses an arrangement for detecting conductive bodios in which not the main oscillation is observed but an oscillation harmonic to it. This is obtained by superposing an auxiliary oscillation which is harmonic to the fundamental one.

Claims allowed - 3.

(271) EINRICHTUNG ZUM NACHWEIS VON LEITENDEN KÖRPERN

(ARRANGEMENT FOR DETECTING CONDUCTIVE BODIES)

Rudolf Hans Richter of Leipzig-Gohlis

398,267 Patent issued July 5, 1924.

The investion described in this patent relates to a method in which, by means of arrangements between the turns of the coil used, or between the coils and the earth, such a capacity contact is obtained that changes in the effective capacity caused by approaching of human and animal bodies, as well as changes caused by other sources of disturbance, may become very small and practically harmless.

Claims allowed - 3.

(272) VERFAHREN ZUR AUSMESSUNG MAGNETISCHER WECHSELFELDER

(METHOD FOR MEASURING MAGNETIC ALTERNATING FIELDS)

"Erda" Aktien Gesselschaft in Göttingen

400,148 Patent issued Aug. 7, 1924.

In this patent a method for measuring magnetic alternating fields, especially for the purpose of ground investigation, is described, in which the revolving axis of a coil turned around slowly is adjusted so that the variations of the alternating current induced in the coil by the earth currents,

which vary in the field of the lines of force during rotation, disappear.

Claims allowed - 4.

(273) VERFAHREN ZUR ERFORSCHUNG DES ERDINNERN MITTELS EINES VON EINEM LUFTSCHIFF ODER FLUGZEUG ÜBER DEN BODEN GEFÜHRTEN OFFENEN ODER GESCHLOSSENEN SCHWINGUNGSKREISES

(METHOD FOR INVESTIGATING THE SUBSOIT BY MEANS OF AN OPEN OR CLOSED OSCILLATORY CIRCUIT CARRIED ALONG ABOVE THE GROUND BY AN AIRSHIP OR AIRPLANE)

Dr. Heinrich Löwy of Vienna

401,443 Patent issued Sept. 4, 1924.

This patent relates to a method characterized by an arrangement such that the oscillatory circuit is not drawn along on the ground during the flight but is secured at a constant distance below the airship or airplane and is thus moved above the ground.

Claims allowed - 2.

(274) SCHALTUNGSANORDNUNG FÜR ERDERFORSCHUNG

(ELECTRICAL CIRCUIT ARRANGEMENTS FOR EARTH INVESTIGATION)

"Erda" Aktien Gesellschaft und Dr. Richard Ambronn of Göttingen

403,213 Patent issued Oct. 12, 1924.

This method of connection for earth investigation by means of electrical currents is characterized by producing, on an area situated outside of the region to be investigated, a tension field of a form similar to that of the field. The spatial or superficial distribution of the tension with regard to this field is then measured directly or with the aid of a potentiometer inserted in this tension field.

Claims allowed - 2.

(275) VERFAHREN ZUR ELEKTRODYNAMISCHEN ERFORSCHUNG DES ERDINNERN MITTELS DIE ERDSCHICHTEN DURCHSTRAHLENDER, VON TELEPHONEMPFÄNGERN AUFZUFANGENDER ELEKTRISCHER INDUKTIONSSTRÖME

(METHOD FOR ELECTRO-DYNAMIC INVESTIGATION OF THE SUBSOIL BY MEANS OF ELECTRICAL INDUCTION CURRENTS PENETRATING THROUGH THE EARTH-LAYERS AND CAUGHT BY TELEPHONE RECEIVERS)

Nobert Gella of Vienna

403,494 Patent issued Oct. 2, 1924.

The method for investigating the subsoil disclosed in this patent consists in leaving the receiving station at the same place and moving the

transmitter along the borders of the region under investigation, determining the strength of the sound by means of a telephone.

Claims allowed - 2.

(276) EINRICHTUNG ZUM NACHWEIS UND ZUR MESSUNG DES ABSTANDES ELEKTRISCH LEITFÄHIGER MASSEN

(METHOD FOR DETERMINING AND MEASURING THE DISTANCE FROM ELECTRICALLY CONDUCTIVE MASSES)

Dr. Heinrich Löwy of Vienna

403,939 Patent issued Oct. 7, 1924.

The method for determining and measuring the distance from electrically conductive masses described in this patent is characterized by an arrangement in which a transmitter and a receiver are switched off alternatively and with variable frequency by means of a modulation transmitter (Modulationssender). The modulation is carried out by alternating current, electrical oscillations, direct current with reduced pulsations, or electrical vibrations.

Claims allowed - 3.

(277) VERFAHREN ZUR RICHTUNGSBESTIMMUNG VON ELEKTROMAGNETISCHEN STRAHLUNGEN, VORZUGSWEISE IN BERGWERKEN

(METHOD FOR DETERMINING THE DIRECTION OF THE ELECTROMAGNETIC RADIATION ESPECIALLY IN MINES)

"Erda" Aktien-Gesellschaft of Göttingen

404,098 Patent issued Oct. 11, 1924.

The method for determining the direction of the electromagnetic radiation described in this patent consists of determining, by means of a frame antenna, the direction of magnetic lines of force produced by two transmitting antennae, located at the same place or at two places close one to another, the magnetic axes of which are perpendicular one to another and also perpendicular to the direction toward the receiving station.

Claims allowed - 3.

(278) EINRICHTUNG ZUM NACHWEIS VON LEITENDEN ODER MAGNETISIERBAREN KÖRPERN

(ARRANGEMENT FOR DETERMINING CONDUCTIVE OR MAGNETIZABLE BODIES)

Rudolf Hermann Heinrich Geffcken and Rudolf Hans Richter
of Leipzig-Gohlis

404,099 Patent issued Oct. 13, 1924.

This patent discloses an arrangement for determining conductive or magnetizable bodies by means of frequency changes which are made perceivable in the form of changes of sound frequency oscillations. The arrangement is characterized by determining the frequency changes of the sound frequency oscillations by working with other oscillations or other systems of oscillations.

Claims allowed - 6.

(279) EINRICHTUNG ZUR AUFSUCHUNG UND BEGRENZUNG ELEKTRISCH LEITENDER RÄUME IN RÄUMEN VON ABWEICHENDER LEITFÄHIGKEIT ODER DIELEKTRIZITÄTSKONSTANTE

(METHOD FOR DETECTING AND DELIMITING ELECTRICALLY CONDUCTIVE SPACES SURROUNDED BY SPACES WITH ANOMALOUS CONDUCTIVITY OR DIELECTRIC CONSTANT)

"Erda" Aktien-Gesellschaft in Göttingen

405,473 Patent issued Nov. 1, 1924.

This patent discloses a method by which electrically conductive spaces surrounded by spaces with anomalous conductivity or dielectric constant are detected and their borders are established by determining the direction and intensity of electrical waves emitted from a fixed place and deflected from the surface limiting the conductive space and running along suitably situated lines around the surface sought for.

Claims allowed - 1.

(280) VERFAHREN ZUR UNTERSUCHUNG DER BODENBESCHAFFENHEIT AUF GRUND DER VERTEILUNG DER ELEKTRISCHEN LEITFÄHIGKEIT IM BODEN

(METHOD FOR INVESTIGATION OF THE NATURE OF THE GROUND BASED ON THE DISTRIBUTION OF ELECTRICAL CONDUCTIVITY IN THE SOIL)

"Erda" Aktien-Gesselschaft in Göttingen

406,299 Patent issued Nov. 17, 1924.

The method described in this patent consists in measuring and comparing the variations of the natural magnetic earth field produced by natural earth currents as well as artificial current impulses. The measurements are made at several points, suitably distributed, by means of a magnetometer or adjusted resonance systems.

Claims allowed - 3.

(281) VERFAHREN ZUM FESTSTELLEN VON WASSERANNÄHERUNG BEI VERROHRTEN BOHRUNGEN

(METHOD FOR DETERMINING THE APPROACH OF WATER IN TUBED BOREHOLES)

Gesellschaft für praktische Geophysik, m.b.H. in Freiburg.

407,495 Patent issued Dec. 17, 1924.

The method described in this patent is characterized by an arrangement in which the metallic tubing of a borehole itself forms the capacity of a resonance current; the tubing capacity and its change is measured directly in a known way, either by the measurement of the charge current or the discharge current, or by using a combination with another resonance current according to the well-known method of electrical vibrations; the capacity change of the tubing caused by the approach of water is determined from the change in the rate of vibration of the sound or from the strength of the charge and the discharge current.

Claims allowed - 2.

(282) VORRICHTUNG ZUM ENTSENDEN ELEKTRISCHER STRÖME IN DEN BODEN ZWECKS AUFSUCHUNG UND LAGENBESTIMMUNG VON BODENTEILEN MIT ABWEICHENDER ELEKTRISCHER LEITFÄHIGKEIT

(ARRANGEMENT FOR DISPATCHING ELECTRICAL CURRENTS INTO THE GROUND FOR THE PURPOSE OF DISCOVERING AND DETERMINING THE POSITION OF DEPOSITS WITH ANOMALOUS ELECTRICAL CONDUCTIVITY)

W. Piepmeyer and Co., Kommandit-Gesellschaft in Cassel-Wilhelmshöhe

408,819 Patent issued Jan. 27, 1925.

This patent discloses an arrangement in which insulated electrodes are buried in the ground at a considerable depth. For this a pipe, the inside surface of which is covered with an insulating layer, is driven into the ground. An insulated cable passes through the pipe and is connected with the metallic sharp end of the pipe; the connection with the ground is established through this metallic end which is insulated from the pipe.

Claims allowed - 2.

(283) VERFAHREN ZUM AUFSUCHEN UND ZUR LAGEBESTIMMUNG VON BODENTEILEN MIT ABWEICHENDER ELEKTRISCHER LEITFÄHIGKEIT MIT HILFE IN DEN BODEN GESANDTER ELEKTRISCHER STRÖME

(METHOD FOR DISCOVERING AND DETERMINING THE POSITION OF PARTS OF THE GROUND HAVING ANOMALOUS ELECTRICAL CONDUCTIVITY BY MEANS OF ELECTRICAL CURRENTS DISPATCHED INTO THE GROUND)

W. Pipemeyer and Co., Kommandit-gesellschaft in Cassel-Wilhelmshöhe

415,188 Patent issued June 22, 1925.

The plan of this patent is characterized by measuring the direction and strength of the magnetic field produced at places above the surface of the ground or those accessible in the subsoil, in such a way that a wire-wound ring, capable of being turned in any position by means of three axes, is connected to a measurement device; for the determination of the direction of the magnetic field first one axis, which at the same time is the diameter of the ring, is brought into such a position that by rotating the ring around this axis the device for measuring the current indicates the absence of a current at any position of the ring; after this the maximum value of the induction is determined for the position of the ring-plane perpendicular to this position of the axis.

Claims allowed - 2.

(284) VERFAHREN ZUM AUFFINDEN LEITENDER MASSEN IM ERDBODEN DURCH ELEKTRISCHEN WECHSELSTROM, DER AN ZWEI STELLEN DEM BODEN ZUGEFÜHRT WIRD

(METHOD FOR DISCLOSING CONDUCTIVE SUBTERRANEAN MASSES BY MEANS OF ELECTRIC ALTERNATING CURRENT PASSED INTO THE GROUND AT TWO POINTS)

Société Industrielle des Procédés W.-A. Loth in Paris

417,663 Patent issued Aug. 14, 1925.

Determination is made of the earth-currents produced by means of a line, the ends of which are connected with the ground, by means of their induction-effect upon a coil. The particular properties of the ground are disclosed by the course of the magnetic lines of force. Cardan suspension of the coil characterizes this method. A system of three frames adjusted perpendicularly one to another is used.

Claims allowed - 4.

(285) EINRICHTUNG ZUM NACHWEIS VON LEITENDER ODER MAGNETISCHEN KÖRPERN

(ARRANGEMENT FOR DETERMINING CONDUCTIVE OR MAGNETIC BODIES)

Dr. Rudolf Hermann Heinrich Geffcken and Dr. Rudolf Hans Richter of Leipzig-Gohlis

417,834 Patent issued Sept. 19, 1925.

The arrangement for determining conductive or magnetic bodies based on their influence upon the vibration conditions of a tubular generator is characterized by determining the changes of the anode direct current or grid direct current of the generator tubes caused by this influence.

Claims allowed - 5.

(286) VERFAHREN ZUR ELEKTRISCHEN UNTERSUCHUNG DES UNDERGRUNDES

(METHOD FOR ELECTRICAL INVESTIGATION OF THE SUBSOIL)

"Erda" Aktien-Gesselschaft of Göttingen

422,348 Patent issued Nov. 28, 1925.

High-frequency alternating currents are passed into the ground at two or more suitable selected points and the intensity, direction, and the polarization conditions of these alternating currents, at points suitably distributed, are determined by means of inclosing in the electromagnetic alternating field produced by the current-receiving systems for high-frequency oscillations.

Claims allowed - 2.

(287) EINRICHTUNG ZUM ELEKTROAVIATISCHEN NACHWEIS UND ZUR MESSUNG DES ABSTANDES ELEKTRISCH LEITFÄHIGER MASSEN

(ARRANGEMENT FOR ELECTRO-AVIATIC INVESTIGATION AND FOR MEASURING DISTANCES FROM ELECTRICALLY CONDUCTIVE MASSES)

Dr. Heinrich Löwy of Berlin

425,050 Patent issued Feb. 11, 1926.

The arrangement described in this patent consists of connecting two or more airships by metallic wires serving as an antenna.

Claims allowed - 4.

(288) EINRICHTUNG ZUM NACHWEIS VON LEITENDEN KÖRPERN

(ARRANGEMENT FOR DETERMINING CONDUCTIVE BODIES)

Dr. Rudolf Hermann Heinrich Geffcken and
Dr. Rudolf Hans Richter
of Leipzig-Gohlis.

426,795 Patent issued March 17, 1926.

According to the arrangement described in this patent the determination of conductive bodies is obtained by means of several loosely coupled systems, capable of oscillating and adjusted to the exciting frequency in such a manner that the oscillation produced in each system remains, with regard to the inducing oscillation, in the region of sharp phase change; an arrangement for determining the shifts of the phase conditions of the systems is added.

Claims allowed - 3.

(289) EINRICHTUNG ZUM NACHWEIS ELEKTRISCH LEITFÄHIGER SCHICHTEN DES ERDBODENS

(ARRANGEMENT FOR DETERMINING ELECTRICALLY CONDUCTIVE LAYERS OF THE GROUND

Dr. Heinrich Löwy of Vienna

432,221 Patent issued July 27, 1926.

This patent describes the determination of electrically conductive layers of the ground by means of an electrical oscillator carried by an airship; the invention is characterized by the arrangement in which the metallic frame of the airship serves as an antenna.

Class allowed - 1.

(290) EINRICHTUNG ZUM NACHWEIS VON LEITENDEN ODER MAGNETISIERBAREN KÖRPERN DURCH FREQUENZÄNDERUNG VON SCHWINGUNGEN

(ARRANGEMENT FOR DETERMINING CONDUCTIVE OR MAGNETIZABLE BODIES BY CHANGE OF FREQUENCY OF OSCILLATIONS)

Rudolf Hermann Heinrich Geffcken and Rudolf Hans Richter
of Leipzig-Gohlis.

435,255 Patent issued Oct. 11, 1926.

At least two oscillation systems are coupled one with another and adjusted in such a manner that in case of changes of natural frequency of one system, stronger changes of the coupled frequencies or of the frequency of one of the really existing oscillations appear. It is essential that more than one of the coupled oscillation systems be provided with a generator.

Claims allowed - 5.

(291) ANORDNUNG DER ELEKTRODEN FÜR EIN ELEKTRISCHES STROMVERFAHREN ZU BODENUNTERSUCHUNGEN

(ARRANGEMENT OF ELECTRODES FOR AN ELECTRICAL CURRENT METHOD FOR INVESTIGATING THE SOIL)

Käte Heckmann of Freiburg i. B.

435,364 Patent issued Oct. 11, 1926.

The invention described in this patent recommends the arrangement of several primary electrodes along one line at equal distances one from another; the signs of the electric tension given them must alternate; the secondary electrodes are distributed on both sides of each primary electrode so that their tension difference is equal to zero.

Claims allowed - 1.

(292) EINRICHTUNG ZUR MESSUNG UND ZUM NACHWEIS KLEINER ÄNDERUNGEN DER FREQUENZ VON ELEKTRISCHEN SCHWINGUNGSKREISEN, INSBESONDERE ZUM NACHWEIS DER DURCH UNTERIRDISCHE ELEKTRISCHE LEITER VERURSACHTEN ÄNDERUNGEN DER FREQUENZ VON LUFTFAHRZEUG-ANTENNEN

(ARRANGEMENT FOR MEASURING AND DETERMINING SMALL FREQUENCY CHANGES IN ELECTRIC RESONANCE CURRENTS, ESPECIALLY FOR DETERMINING THE FREQUENCY CHANGES IN THE AIRSHIP-ANTENNAE CAUSED BY SUBTERRANEAN ELECTRIC CONDUCTORS)

Dr. Heinrich Löwy of Vienna

436,827 Patent issued Nos. 13, 1926.

The fundamental part of the invention described in this patent discloses a method by which the electrical oscillation system under investigation is arranged in such a manner (for example, in case of tubular senders, by means of suitable choice of the constants of the oscillation current) that the oscillation process is displaced to the close proximity of a place of rupture or to the region of a sharp drop of the oscillation amplitude in which great changes of the oscillation amplitude correspond to small changes of frequency.

Claims allowed - 1.

(293) VERFAHREN ZUR BESTIMMUNG DER TIEFE VON EINLAGERUNGEN IM ERDBODEN

(METHOD FOR DETERMINING THE DEPTH OF DEPOSITS IN THE GROUND)

"Seismos" Gesellschaft m.b.H. of Hanover

440,106 Patent issued Jan. 29, 1927.

This patent discloses a method for determining the depth of deposits from the deformation of the field of lines of force. A straight cable is put

over the ground and covered with earth on both ends. The current field is studied by means of secondary electrodes. The distance between the electrodes can be increased gradually; the electrodes may be disposed symmetrically or asymmetrically with respect to the source of the current which is placed in the center of them. By measuring the electric field for every position of the electrodes, a sudden and strong change of this field will indicate the presence of the deposit and will make the determination of its depth possible.

Claims allowed - 1.

(294) VERFAHREN ZUR FESTSTELLUNG UND LOKALISIERUNG VON KÖRPERN IM UNTERGRUNDE

(METHOD FOR DETERMINING AND LOCATING SUBTERRANEAN BODIES)

"Seismos" Gesellschaft m.b.H. of Hannover

442,832 Patent issued April 8, 1927.

This patent describes a method for determining and locating subterranean bodies, the electrical conductivity and the dielectric constant of which are anomalous with respect to masses surrounding them. The investigation is carried out by measuring the phase difference at various points on the ground by means of electrodes. Claims allowed - 2.

(295) VERFAHREN ZUR AUFSUCHUNG VON EINLAGERUNGEN IM ERDBODEN

(METHOD FOR LOCATING SUBTERRANEAN DEPOSITS)

Dr. Johann Königsberger of Freiburg i. B.

444,506 Patent issued May 21, 1927.

Deposits (iron ore, water, coal) which are surrounded entirely or partly by substances of a lower electrical conductivity or have a lower dielectric constant are located by measuring at various receiving stations placed approximately at equal distances from a sending station, electrical waves emitted from this sending station. The invention is based on the theory that better electrical conductivity and greater dielectric constant of bodies cause a certain concentration of waves along the border surfaces of these bodies.

Claims allowed - 1.

(296) VORRICHTUNG ZUR AUSMESSUNG DER MAGNETISCHEN KRAFTLINIEN EINES ELEKTROMAGNETISCHEN WECHSELFELDES, VORZUGSWEISE ZU ERDERFORSCHUNGSZWECKEN

(APPARATUS FOR MEASURING MAGNETIC LINES OF FORCE OF AN ELECTROMAGNETIC ALTERNATING FIELD, EXPECIALLY FOR THE PURPOSE OF GROUND INVESTIGATION)

Dr. Richard Ambronn of Göttingen

448,190 Patent issued Aug. 6, 1927.

This patent discloses an apparatus for measuring magnetic lines of

force of an electromagnetic alternating field which is characterized by the following arrangement: The position of the plans of turn of the induction coil in space is indicated by a pendulum body provided with a vertical galvanoscope; this pendulum is suspended so that it can be moved freely around two axes perpendicular one to another and situated in the same plane; one of these axes is perpendicular to the plane of turn of the induction coil.

Claims allowed - 1.

(297) VERFAHREN UND VORRICHTUNG ZUR BESTIMMUNG VON ÄQUIPOTENTIALFLÄCHEN UND-LINIEN

(METHOD AND APPARATUS FOR DETERMINING EQUIPOTENTIAL SURFACES AND EQUIPOTENTIAL LINES)

Hermann Heuer of Gandersheim, Harz

453,564 Patent issued Dec. 14, 1927.

A method and apparatus is presented for measuring equipotential surfaces and equipotential lines by which their determination becomes more accurate owing to the possibility of compensating and measuring the undesirable induction currents.

Claims allowed - 5.

(298) VERFAHREN UND VORRICHTUNG ZUM MESSEN VON ÄQUIPOTENTIALFLÄCHEN UND-LINIEN

(METHOD AND APPARATUS FOR MEASURING EQUIPOTENTIAL SURFACES AND EQUIPOTENTIAL LINES)

Hermann Heuer of Gandersheim, Harz

458,021 Patent issued March 31, 1928.

This patent is an addition to patent 453,564 concerning the method and apparatus for measuring equipotential surfaces and equipotential lines. The present patent discloses an invention for discovering spatial distribution of current by means of two measurement currents, each provided with two probes, branched from the spatial conductor--for example, the ground.

Claims allowed - 7.

(299) VERFAHREN ZUR BESTIMMUNG DES ABSTANDES EINER REFLEKTIERENDEN FLÄCHE MITTELS ELEKTROMAGNETISCHER WELLEN, INSBESONDERE ZUR FESTSTELLUNG DER LAGE ELEKTRISCH LEITENDER FLÄCHEN IN DER ERDE

(METHOD FOR DETERMINING THE DISTANCE FROM A REFLECTION SURFACE BY MEANS OF ELECTROMAGNETIC WAVES, ESPECIALLY FOR DETERMINING THE POSITION OF ELECTRICALLY CONDUCTIVE SURFACES IN THE GROUND)

"Seismos" G. m.b.H. of Hannover

458,097 Patent issued April 4, 1928.

The method discussed in this patent is characterized by an arrangement in which the deviation of waves of different lengths emitted from a fixed sending station and received by a fixed frame antenna, serving as the receiving station, is investigated in connection with the phase difference.

Claims allowed - 3.

(300) EINRICHTUNG ZUR BESTIMMUNG DER RÄUMLICHEN LAGE DER LUFTELEKTRISCHEN ÄQUIPOTENTIALFLÄCHEN UND POTENTIALDIFFERENZEN, INSBESONDERE FÜR DIE ZWECKE DER ELEKTRISCHEN BODENERFORSCHUNG

(ARRANGEMENT FOR DETERMINING THE SPATIAL POSITION OF AERIAL ELECTRICAL EQUIPOTENTIAL SURFACES AND POTENTIAL DIFFERENCES, ESPECIALLY FOR THE PURPOSE OF ELECTRICAL INVESTIGATION OF THE GROUND)

Gebr. Ruhstrat of Göttingen

461,908 Patent issued July 2, 1928.

Two collectors, rotatable vertically and horizontally about corresponding disks and connected by a line with an electrometer are placed in such a manner that the inclination of the line connecting the two collectors toward a horizontal plane, its horizontal direction, and the potential difference between the two collectors can be read directly from the apparatus.

Claims allowed - 3.

(301) VERFAHREN ZUR BODENERFORSCHUNG AUF ELEKTROMAGNETISCHEM WEGE, BEI WELCHEM DEM BODEN WECHSELSTROM GALVANISCH ZUGEFÜHRT UND DIE STÄRKE DES VON IHM ERZEUGTEN MAGNETISCHEN FELDS MIT HILFE EINER INDUKTIONSSPULE ERMITTELT WIRD

(METHOD FOR ELECTROMAGNETIC INVESTIGATION OF THE GROUND IN WHICH THE ALTERNATING CURRENT IS PASSED INTO THE GROUND GALVANICALLY AND THE STRENGTH OF THE MAGNETIC FIELD PRODUCED BY IT IS DETERMINED BY MEANS OF AN INDUCTION COIL)

Svenska Radioaktiebolaget of Stockholm

463,245 Patent issued July 26, 1928.

A method is described in which the electromagnetic investigation of

the ground is carried out with the aid of an induction coil. The potential difference produced in the coil by induction is measured with a tubular voltmeter.

Claims allowed - 1.

(302) VERFAHREN ZUM AUFSUCHEN WERTVOLLER BODENEINLAGERUNGEN AUF ELEKTROMAGNETISCHEM WEGE DURCH ZUFÜHRUNG ELEKTRISCHER WECHSELSTRÖME

(METHOD FOR DISCOVERING VALUABLE DEPOSITS ELECTROMAGNETICALLY BY INTRODUCING INTO THE GROUND ELECTRICAL ALTERNATING CURRENTS)

Dr. Richard Ambronn of Göttingen

464,767 Patent issued Dec. 4, 1928.

This patent discloses a method for locating valuable deposits electromagnetically by introducing into the ground alternating current and investigating the corresponding components produced of the magnetic field vector with regard to their mutual relation, as well as their relation to a normal phase.

Claim allowed - 7.

(303) VERFAHREN ZUR PRAKTISCH VOLLSTÄNDIGEN ELIMINIERUNG DES EINFLUSSES DER INDUKTION DER ZULEITUNGEN ZU DEN ELEKTRODEN AUF DAS AUFNAHMEGERÄT EINES ZUM ZWECKE DER BODENERFORSCHUNG IN DEN UNTERGRUND GESANDTEN WECHSELSTROMS

(METHOD FOR PRACTICALLY COMPLETE ELIMINATION OF THE INFLUENCE OF THE INDUCTION OF THE WIRE CONNECTED TO ELECTRODES UPON THE RECEIVING APPARATUS OF AN ALTERNATING CURRENT INTRODUCED INTO THE SUBSOIL FOR THE PURPOSE OF THE INVESTIGATION OF THE GROUND)

Exploration Bodenuntersuchungs- and Verwertungs G. m.b.H. of Berlin.

469,018 Patent issued Nov. 29, 1928.

This patent discloses a method characterized by the use of double wound wires. The first electrode is connected with the generator directly; the wire leading to the second electrode is branched, at a point close to the first electrode, into two lines of equal resistance; these two lines are conducted around the region to be investigated in a symmetrical form (for example in the form of a rectangle). By this arrangement the influence of the induction produced by single parts of the line upon the receiving apparatus is practically eliminated.

Claims allowed - 2.

(304) VERFAHREN ZUR BODENFORSCHUNG AUF ELEKTROMAGNETISCHEN WEGE DURCH ERREGUNG ELEKTRISCHER WECHSELSTRÖME IM UNTERGRUNDE UND AUFNAHME DER ELLIPTISCHEN SCHWINGUNGEN DES BODENSTROMES BZW. DES MAGNETISCHEN VEKTORS

(METHOD FOR ELECTROMAGNETIC INVESTIGATION OF THE GROUND BY ALTERNATING CURRENT PRODUCED UNDER THE GROUND AND DETERMINATION OF THE ELLIPTICAL OSCILLATION OF THE EARTH CURRENT OR OF THE MAGNETIC VECTOR)

Dr. Richard Ambronn of Göttingen

469,445 Patent issued Dec. 17, 1928.

The method described in this patent is characterized by the following proceeding: in all points of observation of the whole region under investigation or of single parts of it those diameters of the ellipses of oscillation are determined for which the field vectors taken in their directions are of the same phase.

Claims allowed - 6.

(305) VERFAHREN ZUR AUSMESSUNG DES FELDES EINES INSBESONDERE ZUM ZWECKE DER BODENERFORSCHUNG MIT ELEKTRODEN IN DEN UNTERGRUND GESANDTEN WECHSELSTROMES

(METHOD FOR MEASURING THE FIELD OF AN ALTERNATING CURRENT SENT UNDER THE GROUND BY MEANS OF ELECTRODES, ESPECIALLY FOR THE PURPOSE OF GROUND INVESTIGATION)

(Addition to patent 453,564)
Hermann Heuer of Gandersheim, and others

470,231 Patent issued Jan. 10, 1929.

This patent, which is an addition to patent 453,564, discloses a simplification of the apparatus. The resonance device is placed, instead of in a special measurement probe line, directly in the main line, and the changes necessary to compensate disturbances serve directly for the location of the disturbing local deposits.

Claims allowed - 2.

(306) VERFAHREN ZUR ELEKTRISCHEN BODENFORSCHUNG DURCH VOLLSTÄNDIGE AUSMESSUNG DER ELEKTRISCHEN UND MAGNETISCHEN ELEMENTE DES DURCH DEN UNDERGRUND ZUGEFÜHRTEN WECHSELSTROME ERZEUGTEN, ELLIPTISCH POLARISIERTEN ELEKTROMAGNETISCHEN FELDES

(METHOD FOR ELECTRICAL INVESTIGATIONS OF THE GROUND BY COMPLETE MEASUREMENT OF ELECTRIC AND MAGNETIC ELEMENTS OF AN ELLIPTICALLY POLARIZED ELECTROMAGNETIC FIELD PRODUCED BY ALTERNATING CURRENTS CONDUCTED THROUGH THE SUBSOIL)

Dr. Richard Ambronn of Göttingen

471,636 Patent issued Feb. 25, 1929.

The method described in this patent discloses an arrangement by which the disturbing effects of vagabond currents are nullified by grounding that point (neutral point) of the measuring apparatus with regard to which the disturbances compensate each other.

Claims allowed - 5.

(307) EINRICHTUNG ZUR BESTIMMUNG DER LAGE VON LUFTELEKTRISCHEN ÄQUIPOTENTIALFLÄCHEN ZUM ZWECKE DER ELEKTRISCHEN BODENFORSCHUNG

(APPARATUS FOR DETERMINING THE POSITION OF ELECTRICAL EQUIPOTENTIAL SURFACES FOR THE PURPOSE OF ELECTRICAL INVESTIGATION OF THE GROUND)

Ernst Ruhstrat of Göttingen

475,434 Patent issued July 11, 1929.

This patent discloses a method for electrical investigation of the ground characterized by application of a highly sensitive electrometer, one part of which, preferably its case, serves as a collector.

Claims allowed - 1.

(308) VERFAHREN ZUR ERMITTELUNG DER LAGE UND GESTALT VON NUTZBAREN LAGERSTÄTTEN ODER GESTEINSSCHICHTEN MITTELS BEOBACHTUNG VON SCHALLWELLEN

(METHOD FOR DETERMINING THE POSITION AND THE SHAPE OF USEFUL DEPOSITS OR ROCK STRATA BY OBSERVING SOUND WAVES)

Piepmeyer and Co., Kommandit-Gesellschaft of Kassel-Wilhelmshöhe

479,724 Patent issued July 22, 1929.

This patent discloses a method for determining the position and the shape of deposits by means of observing the sound waves propagated inside of the earth. The method is characterized by variation of the intensity of the

sound waves in suitable time intervals so that any desired number of time intervals can be observed in various receiving stations. By corresponding shifting of the receiving stations the velocity of propagation to be compared can be determined in the air as well as in the earth strata at various depths; the position and the shape of these deposits can be established accordingly.

Claims allowed - 1.

(309) KAMMSONDE FÜR DIE ZWECKE DER ELEKTRISCHEN BODENERFORSCHUNG

(PROBE COMB FOR THE PURPOSE OF ELECTRICAL INVESTIGATION OF THE GROUND)

Oberschlesische Werkstätten für Präzisionsmechanik
G. m.b.H. of Ratibor, O.-S.

481,087 Patent issued Oct. 24, 1929.

The method described in this patent consists of determining the equipotential lines of a field created by an alternating current by using, instead of one single probe, a probe comb consisting of several metallic rods arranged at small distances on a rod so that any one of them can be connected to the measuring apparatus by means of a slide.

Claims allowed - 3.

(310) VERFAHREN ZUR BESTIMMUNG VON ÄQUIPOTENTIALFLÄCHEN UND-LINIEN, BEI DEM WECHSELSTROM DEM RÄUMLICHEN LEITER ZUGEFÜHRT UND MIT SONDEN ENTNOMMEN WIRD

(METHOD FOR DETERMINING THE EQUIPOTENTIAL SURFACES AND EQUIPOTENTIAL LINES IN WHICH ALTERNATING CURRENT IS LED IN IN THE SPATIAL CONDUCTOR AND IS TAKEN OUT WITH PROBES)

(Addition to patent 453,564)

Hermann Heuer of Gandersheim, and others

485,286 Patent issued Oct. 30, 1929.

This patent discloses a further development of the method for determining equipotential surfaces and equipotential lines as described in the original patent, No. 453,564. Here the high resistances are reduced by a device producing a negative resistance and containing vacuum tubes. The intensity of the heating current of the vacuum tubes indicates the value of the resistance to be compensated, especially concerning the conductivity of geological formations.

Claims allowed - 4.

(311) VERFAHREN ZUR ELEKTRISCHEN BODENERFORSCHUNG

(METHOD FOR ELECTRICAL INVESTIGATION OF THE GROUND)

Prospector Institut für praktische Geophysik und Geologie,
Dr. Hülsenbeck and Co. of Frankfurt on the Main.

489,434 Patent issued Jan. 17, 1930.

The method for electrical investigation of the ground described in this patent concerns a proceeding by which electrical momentary waves are conducted to the ground and the progress in space and time of the distribution of the momentary waves is determined at any point of the area under investigation by means of high-frequency oscillographs.

Claims allowed - 3.

(312) VERFAHREN ZUR BESTIMMUNG DES MAGNETISCHEN VEKTORS DER KÜNSTLICH ERZEUGTEN ERDSTRÖME NACH GRÖSSE UND RICHTUNG BEI GEOPHYSIKALISCHEN UNTERSUCHUNGEN MITTELS WECHSELSTRÖME, BEI WELCHEM ALLEIN DIE GRÖSSE UND RICHTUNG DER ERDSTROM KOMPONENTE GEMESSEN WIRD

(METHOD FOR DETERMINING THE INTENSITY AND DIRECTION OF THE MAGNETIC VECTOR OF ARTIFICIALLY PRODUCED ALTERNATING EARTH CURRENTS IN GEOPHYSICAL INVESTIGATIONS BY MEASURING ONLY THE VALUE AND DIRECTION OF THE EARTH-CURRENT COMPONENT)

Prospector Institut für praktische Geophysik und Geologie,
Dr. Hülsenbeck and Co. of Frankfurt on the Main

490,910 Patent issued Feb. 3, 1930.

The method for determining the intensity and direction of the magnetic vector of artificially produced alternating earth currents by measuring only the value and direction of the earth-current component, described in this patent, consists in determining the ratio of the intensities of the horizontal component and the vertical component.

Claims allowed - 1.

(313) VERFAHREN ZUR ELEKTRISCHEN BODENERFORSCHUNG MITTELS DEM UNDERGRUND INDUKTIV ZUGEFÜHRTER WECHSELSTRÖME UND AUSMESSUNG DES ELLIPTISCH POLARISIERTEN MAGNETISCHEN FELDES DURCH SUCHSPULEN

(METHOD FOR ELECTRICAL INVESTIGATION OF THE GROUND BY MEANS OF ALTERNATING CURRENTS PRODUCED IN THE SUBSOIL BY INDUCTION AND MEASUREMENT OF ELLIPTICALLY POLARIZED MAGNETIC FIELD BY MEANS OF SEARCHING COILS)

(Addition to patent 464,767).

Dr. Richard Ambronn of Göttingen

494,831 Patent issued March 27, 1930.

This patent discloses improvements in the method for electrical investigation of the ground, described in the original patent 454,767, which consist in determining the field component produced by the magnetic whirl as conjugated diameter to that diameter of the ellipse of oscillation of the magnetic vector which corresponds to the magnetic field component produced directly at the point of observation by the alternating current. The measurements are repeated at several frequencies of the alternating current chosen so that the results obtained can be extrapolated to zero frequency.

Claims allowed - 2.

(314) VERFAHREN ZUR ELEKTRISCHEN BODENFORSCHUNG

(METHOD FOR ELECTRICAL INVESTIGATION OF THE GROUND)

Dr. Richard Ambronn of Göttingen

498,912 Patent issued May 28, 1930.

This patent discloses a method for electrical investigation of the ground in which alternating currents of very low frequency (0.3 - 10 per second) are employed and the intensity distribution of which in the current field is measured either directly or by compensation. The measuring instrument used is a galvanometer tuned to the frequency of the sender.

Claims allowed - 3.

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3 - The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

ACCIDENT EXPERIENCE AND COST OF ACCIDENTS AT WASHINGTON COAL MINES[1]

By S. H. Ash[2]

DIFFICULTY IN COMPILING ACCIDENT STATISTICS

In arriving at economic losses in coal mining due to accidents, a statistician is confronted with many difficulties on account of the lack of information relative to nonfatal accidents. For comparative purposes the experience of those countries, States, or companies that keep complete records can be drawn upon and an extimate can be arrived at whereby one State can be compared with another, or one mine compared with another in the same district. Such a comparison for the United States has been compiled by the United States Bureau of Mines.

It will be observed in Table 1 that the cost per ton for accidents in the State of Washington for the year 1927 is higher than for any other State, with the exception of Georgia and North Carolina, with a small combined tonnage of 130,089 tons. (The figures for coal production in Table 1 are taken from Bulletin 319, 1930, of the United States Bureau of Mines.)

The reasons why the cost of accidents in Washington might be higher than the average are on account of physical conditions such as steep pitches, faults, igneous intrusions, highly explosive coal-dust, extremely friable coals, presence of gravel and water, very gassy coal beds, rugged topographic conditions, inherently bad roof and floor, thick beds, and heavy cover. The nature of the coal beds requires the use of preparation machinery. The mines are often deep, and topographic conditions are such that long and fast haulage becomes necessary. All of these factors not only tend to cause high cost of production, which is manifested largely in a lower than average daily production, per man, but they also affect to some degree a tendency toward a higher than average cost for accidents.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6529."

2 District engineer, U. S. Bureau of Mines Safety Station, Berkeley, Calif.

Table 1.- Economic losses in coal mining due to accidents, 1927

State	Production	Men killed	Lost-time accidents[1]	Total days lost[2]	Wage loss, dollars[3]	Compensation loss, dollars[4]	Total loss, dollars	Cost
Alabama	19,765,866	93	14,322	1,503,810	7,519,050	663,090	8,182,140	$0.41
Arkansas	1,548,834	9	1,386	145,530	727,650	64,170	791,820	.51
Colorado	9,724,075	54	8,316	873,180	4,365,900	385,020	4,750,920	.49
Georgia and North Carolina	130,089	2	308	32,340	161,700	14,260	175,960	1.35
Illinois	46,848,224	104	16,016	1,681,680	8,408,400	741,520	9,149,920	.19
Indiana	17,935,758	39	6,006	630,630	3,153,150	278,070	3,431,220	.19
Iowa	2,949,622	15	2,310	242,550	1,212,750	106,950	1,319,700	.45
Kansas	3,443,762	9	1,386	145,530	727,650	64,170	791,820	.23
Kentucky	69,123,998	194	29,876	3,136,980	15,684,900	1,383,220	17,068,120	.25
Maryland	2,314,942	5	770	80,850	404,250	35,650	439,900	.16
Michigan	756,763	3	462	48,510	242,550	21,390	263,940	.35
Missouri	3,064,343	8	1,232	129,360	646,800	57,040	703,840	.23
Montana	2,966,638	13	2,002	210,210	1,051,050	92,690	1,143,740	.38
New Mexico ...	2,935,539	18	2,772	291,060	1,455,300	128,340	1,583,640	.54
North Dakota..	1,527,939	2	308	32,340	161,700	14,260	175,960	.11
Ohio	15,799,697	61	9,394	986,370	4,931,950	434,930	5,366,780	.34
Oklahoma	3,318,054	30	4,620	485,100	2,425,500	213,900	2,639,400	.69
Pennsylvania (bituminous).	133,141,639	356	54,824	5,756,520	28,782,600	2,538,200	31,320,800	.23
Tennessee	5,783,367	17	2,618	274,890	1,374,450	121,210	1,495,660	.26
Texas	1,326,385	2	308	32,340	161,700	14,260	175,960	.13
Utah	4,781,480	26	4,004	420,420	2,102,100	185,380	2,287,480	.48
Virginia	12,916,042	44	6,776	711,480	3,557,400	313,720	3,871,120	.30
Washington ...	2,635,062	22	3,388	355,740	1,778,700	156,860	1,935,560	.74
West Virginia.	145,122,447	593	91,322	9,588,810	47,944,050	4,228,090	52,172,140	.36
Wyoming	6,753,656	22	3,388	355,740	1,778,700	156,860	1,935,560	.29
Other States..	149,231	1	-	-	-	-	-	-
Total	517,763,352	1,742	268,268	28,168,140	140,840,700	12,420,460	153,261,160	0.30
Pennsylvania (anthracite).	80,095,564	489	75,306	7,907,130	39,535,650	3,486,580	43,022,230	.54
Grand total	597,858,916	2,230	343,574	36,075,270	180,376,350	15,907,030	196,283,380	0.33
Average (bituminous).	20,710,534	69.7	10,734	1,127,049	5,635,245	496,961	6,132,206	.30

1 Lost-time accidents based on British experience of 154 nonfatal accidents per fatal accident.
2 Time lost per accident, including fatal accidents, based on figures obtained from annual safety competition of the U. S. Bureau of Mines, is 105 working days per accident.
3 Wage loss is calculated at $5 per day.
4 Compensation loss based on National Safety Council estimate of $46 per accident, including fatalities.

Economic conditions and highly competitive markets have given an intermittent career to many Washington mines, which has resulted more or less in a turnover in the personnel of local operating officials. Developmental programs can not be laid out far in advance, and often a situation results that requires an intensive developmental program, which is almost invariably accompanied by an increase in accidents. On the other hand, the mines do not have a large labor turnover at present, but rather an enviable one. The compensation laws are not drastic nor are the specific individual payments unusually high. The wage scale is satisfactory and the mines work more steadily than in many parts of the country. The State mining laws are more complete and progressive than the average. That Washington can and should have a lower accident cost is proved by the performance at some of the mines where supervision and discipline in the interest of safety as well as mining are being pressed.

SAFETY EDUCATION

In discussing the prevention of fatal accidents at the coal mines one Washington State mine inspector says:

> The State has gone about as far as possible in passing laws to safeguard the mine worker. Education of both the mine worker and mine official to "think safety" and avoid danger is the only way accidents can be reduced.

The recent educational work among the rank and file of industrial workers throughout the country has also been fostered among the coal mine workers of Washington to a degree that soon should show results. Educational work in safety in the past has in large part been directed toward the workman only, but education is necessary for all persons engaged in mining, if accident-prevention work is to be a success, and should start with the management.

In making an address recently at a social gathering in the interest of safety at one of the mines in the State, one of the speakers emphasized in the following terms an outstanding fact that is uppermost in the minds of many companies, including the management of the large Washington mines:

> John Morgan, of the State Department of Labor and Industries, also made a splendid address. Among many other interesting remarks was this: "Gentlemen, I have attended many, many meetings in the past, but I want to say this meeting is unique. And what I mean by that is this: it is unique in that man, his welfare and well-being is uppermost. Unique in its underlying motive to make it possible for the breadwinner to return safely to his little wife and kiddies at the end of every workday. No safety campaign can function properly unless it starts in the swivel chair and carries right through to the face in an unbroken chain.[3]

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3 Pacific Coast Bulletin, vol. 10, No. 11, Seattle, Wash., Dec. 1, 1930, pp. 5 and 15.

Education is essentially a product of several factors which result from instruction, experience, and pertinent information. This later phase of education when applied to accident-prevention work includes the subject of accident cost. It is a phase of mining of vital interest to the operator, workmen, and public. They all assume a part of the cost burden in mining, and a wider knowledge of the financial loss of accidents is responsible in large part for the active interest being taken recently that has for its purpose the lessening of accidents.

In setting up safety standards for the operation of hazardous industries, which include coal mining, the laws[4] of the State contain definitions pertinent to education.

FREQUENCY AND SEVERITY OF ACCIDENTS

The frequency with which all types of mine accidents occur is to a large extent a matter of conjecture, as accidents that do not result in time-loss on account of injury or in loss of life are not taken into account, although they may result in a large economic loss. For this reason some companies keep records of all accidents because any one accident may result in injuries, loss of life, and economic losses. When the same attention that is paid to fatal accidents is given to nonfatal accidents which result in injuries, or which do not result in injuries or loss of life, there is likely to be a large reduction in accident cost. The frequency with which lost-time, nonfatal accidents resulting in injuries, and fatal accidents, occur is a criterion of the frequency of all accidents.

It is often noticeable that as the frequency decreases, the severity per accident increases. The severity of accidents is important in throwing light on the economic loss and physical suffering resulting from accidents, as well as giving an idea as to the extent of the injuries and accident cost. Obviously, with no frequency there can be no severity.

In computing the frequency and severity rates, available State data have been used such as are embodied in the reports of the State mine inspector and of the department of labor and industries. The accident-frequency rate is based on the number of fatal and nonfatal, lost-time injuries per million man-hours. The accident-severity rate indicates the number of calendar days of disability from accidents per thousand man-hours worked.[5]

The accident-frequency rate per million hours and the accident-severity rate per thousand hours for 1927, 1928, and 1929 and for the period 1913-1929 are given in Table 2.

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4 Laws of Washington, Sec. 7675, Industrial Insurance Law, 1929.

5 Adams, W. W., Coal Mine Fatalities in the United States, 1928: Bull. 319, Bureau of Mines, 1930, 125 pp.

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

Table 2.- Frequency and severity of accidents at Washington mines during 1913-1929[1]

Item	1927	1928	1929	1913-1929
Total man-hours	[2]5,731,300	[2]5,465,500	5,804,552	140,378,697
Fatalities:[3]				
Underground	21	9	9	342
Surface	1	0	2	29
Total	22	9	11	371
Time lost through fatalities, days	132,000	54,000	66,000	2,226,000
Compensable total permanent disability claims:				
Number	3	3	6	64
Time lost, days[4] ..	18,000	18,000	36,000	384,000
Compensable permanent part disability claims:				
Number	57	83	65	1,313
Time lost, days ...	--	--	18,226	188,153
Temporary disability claims:				
Number	581	585	563	11,143
Time lost, days ...	--	--	14,350	255,572
Total time lost through temporary disability and permanent part disability claims, days	22,641	36,465	32,576	443,725
Total time lost for all accidents, days..	172,641	108,465	134,576	3,053,725
Total fatalities and compensable non-fatalities	663	680	645	12,891
Accident frequency (per million man-hours)[5]	115.6	124.4	111.1	91.8
Accident severity (time lost, days per thousand man-hours)[5]	30.122	19.845	23.184	21.753

1 Compiled from reports of Department of Labor and Industries, State of Washington.

2 Adams, W. W., Coal Mine Fatalities in the United States, 1928: Bull. 319, Bureau of Mines, 1930, p. 54.

3 State Mine Inspector Reports.

4 6,000 days time lost per accident.

5 Adams, W. W., Quarry Accidents in the United States, 1928: Bull. 325, Bureau of Mines, 1930, p. 90.

For the United States as follows:

	1927	1928
Accident frequency:		
In anthracite mines	99.6	88.1
In bituminous mines	97.3	85.0
Accident severity:		
In anthracite mines	9.927	10.918
In bituminous mines	15.739	12.471

The foregoing figures show that the Washington rate is high when compared with either the anthracite or bituminous mines of the country. Although 1929 shows an improvement, the rate is considerably higher than in past years. It is apparent that there is considerable room for improvement, and although accident costs may appear high to some, the real difficulty lies in the frequency with which accidents occur and their severity when they do occur, rather than in excessive compensation rates. It is only by concentrated team work in accident-prevention work that the coal industry in Washington can hope to lower its accident cost.

COST EXPERIENCE OF COMPENSABLE CLAIMS

The Industrial Insurance Law of the State of Washington became effective in 1911, was operative during 1913, and in substance is as follows:

> An act relating to the compensation and medical, surgical and hospital care and treatment and the welfare and safety of workmen engaged in extra-hazardous employments, and to the compensation of the dependents of such workmen in case of death, and to the liability of the employers of workmen so engaged for such compensation and cost of such care and treatment, and to the collection of industrial insurance and medical aid premiums or assessments and fixing the priority thereof, and providing for injunction for nonpayment thereof, and relating to the liability of third parties for accidents occurring to such workmen, and providing for the extension of the benefits of this act to nonextra-hazardous employment.[6]

By the provision of this law the industrial insurance rate in 1929, for coal mining, class 16-1, was 3 per cent of the payroll and the medical aid rate was 8 cents per man-day. The application of the present act as amended dates from and includes the first day of July, 1927.

A compensation schedule in the industrial insurance laws of Washington provides as follows:

> Sec. 7679. Compensation Schedule.
>
> Each workman who shall be injured in the course of his employment, or his family or dependents in case of death of the workman, shall receive out of the accident fund compensation in accordance with the following schedule, and, except as in this act otherwise provided, such payment shall be in lieu of any and all rights of action whatsoever against any person whomsoever.

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6 Remington's Compiled Statutes, Laws of Washington, 1929, Secs. 7673 to 7711.

In defining the status of injuries as a basis for claims, the law contains the following definitions:

(b) Permanent total disability means loss of both legs, or arms, of one leg and one arm, total loss of eyesight, paralysis or other condition permanently incapacitating the workman from performing any work at any gainful occupation.

(f) Permanent partial disability means the loss of either one foot, one leg, one hand, one arm, one eye, one or more fingers, one or more toes, any dislocation where ligaments were severed where repair is not complete, or any other injury known in surgery to be permanent partial disability.......

(k) No workman injured after June 30th, 1923, shall receive or be entitled to receive compensation out of the accident fund for or during the day on which injury was received or the three days following the same.

Table 3, following, is compiled from data furnished by the Department of Labor and Industries of the State of Washington, and shows the industrial insurance cost experience of compensable claims, since 1913, following enactment of the compensation laws in 1911, to and including the year 1929:

Table 3.- Cost experience of compensable claims, 1913-1929

Description	1927	1928	1929	1913-1929
Temporary disability (T.D.) claims:				
Number	581	585	563	11,143
Time loss, days	-	-	14,350	255,572
Average time loss per claim, days	-	-	25.5	22.9
Cost	-	-	$24,246.30	$398,518.03
Average cost per T.D. claim	-	-	$43.07	$35.76
Permanent partial disability (P.P.D.) claims:				
Number	57	83	65	1,313
Time loss, days	-	-	18,226	188,153
Average time loss per claim, days	-	-	280.4	143.3
Total time awards (cost)	-	-	$29,961.95	$292,947.91
Average time award (cost)	-	-	$460.95	$223.11
Amount of permanent disability (P.D.) awards..	-	-	$37,308.00	$600,011.90
Average degree award ...	-	-	$573.95	$456.98
Average total cost per claim	-	-	$1,034.90	$680.09
Total cost of P.P.D. claims	-	-	$67,269.95	$892,959.81
Total cost, T.D. + P.P.D. claims	$67,300.80	$84,547.20	$91,516.25	$1,291,477.84
Total permanent disability (T.P.D.) claims, number...	3	3	6	64

[illegible]

Table 3.- Cost experience of compensable claims, 1913-1929 - Continued

Description	1927	1928	1929	1913-1929
Total cost T.P.D. claims	$17,326.28	$26,336.91	$46,512.70	$384,936.03
Average cost per claim..	$5,775.43	$8,778.97	$7,752.12	$6,014.63
Total cost all nonfatal claims	$84,627.08	$110,884.11	$138,028.95	$1,676,413.87
Cost per nonfatal claim.	$132.02	$165.25	$217.71	$133.89
Fatal claims:				
Number	27	12	15	433
Number requiring pensions	23	7	11	288
Total cost of compensable fatal claims..	$164,571.84	$51,021.36	$63,732.05	$1,341,352.56
Average cost per claim	$7,155.29	$7,288.76	$5,793.82	$4,657.47
Total compensation cost.	$249,198.92	$161,905.49	$201,761.00	$3,017,766.43
Payroll	$5,755,642	$5,159,832	$5,232,214	$109,880,154
Compensation cost per $100 payroll	$4.32	$3.14	$3.86	$2.74
Coal, net tons[1]	2,631,337	2,519,410	2,591,666	50,722,320
Compensation cost, cents per ton	9.5	6.4	7.7	5.95

1 From State Mine Inspector's Reports.

It can be seen from the foregoing table that, although the number of compensable fatal claims was less in 1929 than for the average of the period 1913-1929, the compensation cost per ton has increased from 5.95 cents per ton to 7.7 cents per ton. It will also be observed that the compensation cost per $100 payroll is $1.12 higher than for the average of the period 1913-1929, or when based on the 1929 payroll it represents a loss of $58,600.

The reasons for the increase in accident cost are the increase in frequency and severity of the nonfatal accidents as well as the decrease in tonnage, especially from mines that formerly had a lower accident rate. Table 3 shows plainly the importance of nonfatal accidents and an outstanding reason why more significance should be placed upon them. More publicity should be given to nonfatal accidents through the mine inspector's reports, and more attention should be given to them in safety discussions.

Table 4 is a recapitulation and summary of compensable claims:

Table 4.- Recapitulation and summary of compensable claims

Description	1927	1928	1929	1913-1929
Number of temporary disability (T.D.) and permanent partial disability (P.P.D.) claims	638	668	628	12,456
Time loss T.D. + P.P.D. claims, days	22,641	36,465	32,576	443,725
Gross wage loss (T.D. + P.P.D.)[1]	$157,355	[2]$288,074	$222,494	[1]$3,061,703
Wage loss per day[1]	$6.95	$7.90	$6.83	[1]$6.90
Number of total permanent disability claims (T.P.D.)	3	3	6	64
Number of compensable fatal claims	23	7	11	288
Number of T.P.D. and compensable fatal claims....	26	10	17	352
Total number of compensable claims	664	678	645	12,808
Total compensation cost of claims	$249,198.92	$161,905.49	$201,761,00	$3,017,766.43
Average cost per claim - (all claims)	$375.30	$238.79	$312.81	$235.62
Total number of nonfatal claims (T.D. + P.P.D. + T.P.D.)	641	671	634	12,520
Number of time-loss accidents (T.D. + P.P.D.) per compensable fatal claim..	27.73	95.42	57.09	43.25
Number of total permanent disability accidents per compensable fatal claim..	0.130	0.428	0.546	0.222
Total number man-days[3] ...	784,438	734,156	725,569	17,155,272
Number of nonfatal claims per thousand shifts worked	0.817	0.913	0.873	0.730

1 Gross wage loss. Net wage loss per day: 1927, $4.38; 1928, $4.87; 1929, $4.09.

2 Calculated.

3 State Mine Inspector Reports.

In the foregoing table it can be seen that accident costs are increasing in spite of the fact that fatalities have apparently decreased in 1928 and 1929. It will be observed that 1929 shows a material increase over the 17-year period and 1927, which was not a good year, both in the ratio of nonfatal to fatal compensable claims, and in the number of nonfatal claims per 1,000 shifts worked.

Table 5 summarizes the compensable cases in coal mining for the period 1913-1929:

Table 5.- Review of compensable cases in coal mining for accidents, 1913-1929

Class	Cases		Per cent of total		Cost		Per cent of total	
	1929	1913-1929	1929	1913-1929	1929	1913-1929	1929	1913-1929
Temporary disability (T.D.)	563	11,143	87.3	87.0	$24,246	$ 398,518	12.1	13.2
Permanent partial disability (P.P.D.)	65	1,313	10.1	10.3	67,270	892,960	33.3	29.6
Total permanent disability (T.P.D.)	6	64	0.9	0.5	46,513	384,936	23.1	12.7
Fatal compensable cases	11	288	1.7	2.2	63,732	1,341,353	31.5	44.5
Total number non-fatal claims	634	12,520	98.3	97.8	138,029	1,676,414	68.5	55.5
Grand total	645	12,808	100.0	100.0	201,761	3,017,767	100.0	100.0

The foregoing table shows the compensable cases that have occurred in the coal-mining industry in Washington since the inception of the compensation law. It will be observed that there were 288 compensable claims for lives lost during the period 1913-1929. During this same period 371 persons were killed (Table 2); the difference between this figure and the compensable claims is explained by the fact that 83 of these persons were unmarried and their deaths did not originate pension cases. Their deaths, nevertheless, are a reality, and the number of compensable cases is a variable depending on whether the persons killed were married men or had dependents, or were single men.

It will be observed that there is a striking uniformity in the ratios maintained for the different classes of cases when comparing the number of cases for the year 1929 with the period 1913-1929. The temporary disability and permanent partial disability cases are practically constant, and there is only 0.1 per cent difference between the sum of the total permanent disability and fatal compensable cases in comparing 1929 with the 17-year period. This may be interpreted to indicate that either accident-prevention work, negative or positive, affects all causes alike, or there has been no concentration on any one particular phase of accident occurrence.

The figures also show that, although temporary disabilities constitute 87 per cent of the cases, they account for only 13 per cent of the cost. In occurrence of accidents, Washington's experience is strikingly the same as in other States, notably Arizona, which goes to prove there is nothing vastly different in the percentage of occurrence in one field, or even in metal mines and coal mines, when one is compared with another, although there may be a vast difference in frequency and severity, depending largely on the extent and effectiveness of the accident-prevention work. This relation also shows that too much stress should not be placed on the assumption that high compensation costs are a result of persons making false claims or imposing on the benefits of the law; even if such cases now and then creep in, they do not have an appreciable significance on the total cost. Compared with the total number of cases and cost, such cases are an exception, not a rule. This is shown when it is observed that the cases of permanent partial disability and total permanent disability, although constituting but approximately 11 per cent of the total number of cases, account for approximately 45 per cent of the cost.

On considering the permanent partial disability cases, where there is a loss of either one foot, one leg, one hand, one arm, one eye, one or more fingers, one or more toes, any dislocation where ligaments were severed and repair is not complete, it is observed that they constitute 10 per cent of the total number of cases and represent that class of human wreckage which becomes to a more or less degree a charge of the community. The cost of this type of accident during 1929 was higher than that of fatalities, and on an average constitutes one-third of the total cost. They are chiefly classed as accidents from falls, haulage, and falling or flying coal.

On considering the total permanent disability cases, although they constitute less than 1 per cent of the accidents, they account for 13 per cent of the cost, and in 1929 reached 23.1 per cent of it. Falls of roof are mainly responsible for this type of accident.

The fatal cases, although they account for 44.5 per cent of the cost, constitute only about 2 per cent of the total number of claims. The fact that during the period 1913-1929 there were but 288 cases of the 433 fatal claims (Table 3) that required pensions, emphasizes the necessity of selective employment and accident-prevention work in extra hazardous occupations.

COST OF MEDICAL AID

During the year 1923 there was enacted into law[7] an act providing medical aid for injured workmen. In substance this act is as follows:

7 Remington's Compiled Statutes, Laws of Washington, 1929, Secs. 7712 to 7726.

Sec. 7712. Medical Aid to be Furnished Injured Workmen.

It is the intent to require the industries of the State to furnish medical, surgical and hospital care to their injured workmen and to place the expense thereof upon each industry as near as may be in the proportion in which it produces injury and creates expense.

The rate which each industry shall pay into the medical aid fund shall be as provided in Section 7676 of Remington's Compiled Statutes, which rate may be increased or decreased, by the director of labor and industries through and by means of the division of industrial insurance, based upon the medical aid cost experience of such industry.

From any change made in such rates any employer or workman claiming to be aggrieved may upon application, have a hearing before the division of industrial insurance upon notice to the interested parties and in the manner provided in Section 7697 of Remington's Compiled Statutes, a review by the courts. The body of interested workmen may designate in writing in duplicate, one of them to be the recipient of service upon all of them, one copy to be posted for local convenience, and the other to be filed with the supervisor of industrial insurance. In default of any such designation, service upon any one workman other than the one instituting a complaint shall be service upon all.

Sec. 7713. Creation of Medical Aid Fund.

A fund is hereby created in the state treasury to be known as the medical aid fund. Into it shall be paid by each employer on or before the 15th day of September, 1923, and on or before the 15th day of January, May, and September of each year thereafter for each day's work or fraction thereof done for him in extra-hazardous employment in or during the preceding four calendar months the medical aid rate provided in Sections 7676 and 7712 of Remington's Compiled Statutes.

The employer shall deduct from the pay of each of his workmen engaged in extra-hazardous work one-half of the amount the employer is required by the foregoing provision of this section to pay into said fund for or on account of the employment of such workman.

By the provisions of the law the medical-aid service can be provided either through the State or a contract system approved by the State. The rate deducted for service, however, is established by law and the medical-aid rate in 1929 was 8 cents per day, 4 cents of which was paid by the employer and 4 cents by each employee for each day he worked.

Table 6 shows the medical-aid costs under the non-contract system only, which is handled by the State, together with the actual costs in cents per day under this system:

Table 6.- Actual medical-aid costs - noncontract system, class 16-1, coal mining

Year	Payroll	Medical aid costs	Cost, cents per day
1923	$6,642,892	$38,272.00	6.41
1924	6,261,022	40,753.73	7.25
1925	6,126,229	28,799.98	8.90
1926	5,549,251	10,298.51	5.09
1927	5,755,642	12,008.10	5.03
1928	5,159,832	12,643.38	5.33
1929	5,232,214	11,674.31	4.85
	40,727,082	154,450.01	6.43

NOTE: The variation in the amount of medical-aid costs in different years is materially influenced by the ratio of contract and noncontract systems of medical-aid service. From 1926 the major per cent of this service was under contract system, hence the reduced amount of medical-aid costs from the department medical-aid fund.
No count is made of the number of medical-aid claims by class; hence this information can not be provided.

Prior to the enactment of the medical-aid law, costs had to be assumed in one way or another by the employer and employee, and the law was drafted on the basis of past experience; rates covering the experience since the enactment of the law can well apply to experience prior to the enactment of the medical aid law. Further, the rates under the contract and noncontract systems are practically the same - the difference lying more in the matter of service. On the basis of the records of the medical-aid costs from the Department of Labor and Industries of Washington, Table 7 has been computed to ascertain the medical-aid cost to the coal-mining industry:

Table 7.- Medical-aid costs for the period 1913-1929

Year	At mines on non-contract system: Man-days[1]	At mines on non-contract system: Medical-aid costs	At mines on contract system: Man-days[1]	At mines on contract system: Medical-aid costs	All mines: Total man-days[1]	All mines: Cost per day, cents	All mines: Total cost[1]	All mines: Ratio contract to non-contract[1]
1913-1922					11,606,126	[2]6.43	$ 746,274	-
1923-1929	2,402,021	$154,450	3,147,125	$202,360	5,549,146	6.43	356,810	1.315
1927	238,729	12,008	545,709	27,449	784,438	5.03	39,457	2.287
1928	237,211	12,643	496,945	26,488	734,156	5.33	39,131	2.095
1929	240,707	11,674	484,862	23,516	725,569	4.85	35,190	2.018
1913-1929	-	-	-	-	17,155,272	[3]6.43	1,103,084	-

1 Calculated.
2 Rate for 1923-1929 used.
3 From State Mine Inspectors' Reports.

With the exception of the first year (1923) of the operation of the medical-aid law, no count is or was made of the number of medical-aid claims by class; hence the cost per claim is largely conjectural. However, the records show that during the year 1923 there were 1294 claims that cost $34,022, or $26.29 per claim. Medical aid is given to all injured persons regardless of time loss, and many claims do not indicate compensable cases.

Table 8 shows the approximate cost of medical aid which is required by the industrial insurance law, and indicates that medical costs are decreasing, unquestionably due in large part to first-aid education, especially in the matter of protection against infection:

Table 8.- Relation of medical-aid costs to payroll and output

Year	Total medical aid cost[1]	Payroll[2]		Coal tonnage	
		Total	Cost per $100 of payroll	Total[2]	Cost per ton, cent
1913-1922	$ 746,274	$69,153,072	$1.079	32,271,747	2.3
1923-1929	356,810	40,727,082	0.876	18,450,573	1.9
1927	39,457	5,755,642	0.685	2,631,337	1.5
1928	39,131	5,159,832	0.758	2,519,410	1.5
1929	35,190	5,232,214	0.672	2,591,666	1.4
1913-1929	1,103,084	109,880,154	1.003	50,722,320	2.2

1 Calculated.
2 From Department of Labor and Industries of Washington.

SUMMARY OF ACCIDENT COSTS

The total cost of mine accidents can not be exactly ascertained on account of costs incidental to the rehabitation of properties after certain types of accidents such as explosions, fires, floods, bumps, and others. Disasters are almost invariably followed by a lowering of morale and an increase in labor turnover that are conducive to increased operating costs. Also haulage accidents often result in considerable property damage, frequently without injury to any person.

Table 9 shows approximately the economic losses in coal mining in the State of Washington that have resulted from accidents. It will be observed that the economic losses amount to approximately 50 cents per ton, which is higher than the average for the country. It will also be observed that the daily wage scale is higher than that assumed in Table 1. There was a slight reduction in accident cost for 1929 over 1927, although it is higher for 1929 than the average for the period 1913-1929.

Table 9.- Economic losses at Washington coal mines due to accidents.

Item	1927	1928	1929	1913-1929	United States 1927[4]
Coal production, tons	2,631,337	2,519,410	2,591,666	50,722,320	597,858,916
Men killed	22	9	11	371	-
Total permanent disability claims	3	3	6	64	-
Time lost for temporary disability and permanent part disability accidents, days[1]	22,641	36,465	32,576	443,725	343,574
Time lost for temporary part disability and fatal accidents, days[1].	150,000	72,000	102,000	2,610,000	-
Total time lost, days[2]............	172,641	108,465	134,596	3,053,725	36,075,270
Wage loss due to lost-time accidents[3]	$157,355	$288,074	$222,494	$3,061,703	-
Total wage loss[3]..	$1,199,855	$856,874	$919,154	$21,070,703	$180,376,350
Compensation cost.	$ 249,199	$161,905	$201,761	$ 3,017,766	$ 15,907,030
Medical-aid cost..	$ 39,457	$ 39,131	$ 35,190	$ 1,103,084	-
Total cost.....	$1,488,511	$1,057,910	$1,156,105	$25,191,553	$196,283,380
Cost per ton, cents	57	42	45	49.6	33

1 See Table 2.

2 Based on 6,000 days lost per fatality and each total permanent disability claim.

3 Gross wage loss on basis of $6.95 for 1927 and $6.83 for 1929, as furnished by Department of Labor and Industries of Washington on total disability and permanent part disability claims (average $6.90). Net daily wage loss was $4.38 in 1927, $4.87 in 1928, and $4.09 in 1929.

4 See Table 1.

In arriving at the economic losses due to accidents by causes, it must be remembered that the time loss and compensation costs from each cause are not in direct ratio to the percentage of the number of accidents from the same cause. Statistics for nonfatal accidents have been compiled from the State Mine Inspector's Reports and the resultant economic losses by causes are shown in Table 10:

Table 10.- Economic losses due to accidents, by causes, 1913-1929, showing compensation and medical cost and wage loss

Cause	Fatal and total permanent disability accidents				Nonfatal injuries - temporary disability and permanent part disability				Total		
	Percentage of total number of accidents	Compensation cost	Wage loss cost[1]	Total cost	Percentage of total number of accidents[2]	Compensation and medical-aid cost[3]	Wage loss cost	Total cost	Total cost	Percentage of total cost	Cost per ton of production, cents
Falls of roof and coal....	34.7	$ 600,288	$6,251,500	$6,851,788	36.5	$ 906,402	$1,408,853	$2,315,255	$9,167,043	36.4	18.0
Haulage	13.2	228,000	2,380,380	2,608,380	13.3	509,700	582,000	1,091,700	3,700,080	14.7	7.3
Explosions and fires...	19.8	342,000	3,560,500	3,902,500	2.9	62,000	91,850	153,850	4,056,350	16.1	8.0
Explosives...	3.4	58,000	612,700	670,700	0.3	21,000	21,600	42,600	713,300	2.8	1.4
Electricity..	6.0	103,500	1,080,540	1,184,040	0.4	10,500	10,800	21,300	1,205,340	4.8	2.4
Other causes underground.	17.5	302,000	3,150,500	3,452,500	32.4	833,600	874,600	1,708,200	5,160,700	20.5	10.2
Machinery outside	2.0	34,500	360,180	394,680	1.9	11,400	16,000	27,400	422,080	1.7	0.8
Other causes outside	3.4	58,000	612,700	670,700	6.8	39,960	56,000	95,960	766,660	3.0	1.5
All underground	91.7	1,583,006	16,514,253	18,097,259	88.6	2,091,378	2,724,916	4,816,294	22,913,553	91.0	45.1
All surface..	8.3	143,282	1,494,747	1,638,029	11.4	303,184	336,787	639,971	2,278,000	9.0	4.5
Totals..	100.0	1,726,288	18,009,000	19,735,288	100.0	2,394,562	3,061,703	5,456,265	25,191,553	100.0	49.6

1 Time loss taken on basis of 6,000 days per accident.
2 Compiled from State Mine Inspectors' Reports.
3 Calculated from percentage of compensation paid by Department of Labor and Industries of Washington.

A careful study of the foregoing tables shows that the cost of accidents in coal mining in Washington is such that a considerable saving can undoubtedly be effected by intensive accident-prevention work.

In showing figures that indicate economic losses, statisticians are confronted with a problem which many persons do not comprehend - the fact that loss in wages is a vital cost item; they believe that the only real cost is the compensation cost. Directly the compensation cost is the one the operator must meet, and with equal certainty the loss in wages is a burden that the workmen must sustain. The operator is obliged in one manner or other to assume this cost indirectly if not directly by wage schedules. In any event, the summation of these two items is passed on to the consumer or to the public at large and is a true cost burden that should ultimately be assumed by the industry.

In Table 11 a summary is given of the various items of accident cost that have a direct bearing on economic losses in coal mining:

Table 11.- Summary of accident costs statistics, 1913-1929

Item	1927	1928	1929	1913-1929
Cost per temporary disability accident	-	-	$ 43.07	$ 35.76
Cost per permanent part disability accident	-	-	1,034.90	680.09
Average cost per temporary and permanent part disability accidents	$ 105.48	$ 126.60	145.72	103.68
Cost per total permanent disability accident	5,775.43	8,778.97	7,752.12	6,014.63
Average cost per nonfatal accident	132.02	165.25	217,71	133.89
Average cost per compensable fatal accident	7,155.29	7,288.76	5,793.82	4,657.47
Average cost per compensable accident for all accidents ..	375.30	238.79	312.81	235.62
Nonfatal accidents per 1,000 shifts worked	0.817	0.913	0.873	0.730
Cost per $100 payroll, all accidents:				
Compensation	$4.32	$3.14	$3.860	$2.74
Medical aid	0.685	0.758	0.672	1.003
Wage loss	20.85	16.60	17.560	19.15
Total	25.86	20.50	22.09	22.90
Cost per ton, all accidents, cents:				
Compensation	9.5	6.4	7.7	5.95
Medical aid	1.5	1.5	1.3	2.1
Wage loss	46.0	34.0	36.0	41.5
Total	57.0	42.0	45.0	49.6

It will be observed from the foregoing table that increasing accident costs are due in large part to the increased cost of nonfatal claims. If the years 1928 and 1929 are eliminated, it will be observed from the accident statistics that there is actually a greater number of fatalities for the 5-year period following 1921. The encouraging feature is that 1928 and 1929 show a favorable reduction in fatalities; however, they do not show a favorable reduction in nonfatal accidents. The cost per nonfatal accident in 1929 was $217.71 compared with $133.89 for the period 1913-1929.

A study of the statistics shows the outstanding fact that tons per life lost and man-days per life lost can be favorably reduced and the accident cost still increase, due to the greater frequency and severity of nonfatal accidents. The principal reason for this is that there was a reduction in fatalities due to explosions and electrical contacts. These two causes alone have contributed heavily to accident costs prior to 1928. Accidents caused by explosions and electrical contact are preventable. Although accidents from these causes have constituted a large percentage of the fatalities, they have constituted a very small percentage of the nonfatal accidents, and for this reason the elimination of such accidents does not materially affect the nonfatal accident cost.

It is by the reduction of accidents, both fatal and nonfatal, from falls of roof that accident costs can be materially lessened. It is from this cause that the greatest number of fatal and total permanent disability accidents occur. Their prevention appears to be practically at a standstill; the solution lies in more nearly adequate supervision, more rigid discipline, the strict enforcement of safety rules, and last but not least, the adoption and maintenance of a well thought out system of timbering or other support.